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Safety Aspects of Longwall Mining in the Illinois Coal Basin

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Safety Aspects of Longwall Mining in the Illinois Coal Basin

By Ernest A. Curth

UNITED STATES DEPARTMENT OF THE INTERIOR
Cecil D. Andrus, Secretary

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CONTENTS

	Page
Abstract	1
Introduction	1
Acknowledgments	2
Mine design	2
Premining investigations	3
Roof support designed for hazardous ground	6
Face equipment with safeguards	11
Face installation and personnel training	14
Face operation	17
Panel-to-panel transfer	23
Accident experience	30
Ground control and subsidence study	30
Conclusions	36
References	37

ILLUSTRATIONS

1.	Layout of panels 1 and 2	3
2.	Bearing test apparatus	4
3.	Geologic columns according to cores from boreholes drilled from the surface of mines 21 and 24	5
4.	Key plan of project area	6
5.	Shield during installation	8
6.	Thyssen RHS 18/30 L roof shield	8
7.	Piston-type load indicator with load chart	9
8.	Roof support testing laboratory, test frame for powered supports in the background	10
9.	Roof support plan for gate roads	12
10.	Props in gate road	13
11.	Load monitoring	14
12.	Installation of Eickhoff shearer	15
13.	Three-dimensional model of face operation	16
14.	Longwall mode of operation	18
15.	Shearer at headgate	19
16.	Gob before the first roof fall	20
17.	Shield contact with roof	21
18.	Map of panels 1 and 2 by Illinois State Geological Survey	22
19.	Face advance in terms of time, measurements taken in supply gate	23
20.	Recovery room prior to holing through	24
21.	Props in recovery room	25
22.	Pinrail placed in shale roof	26
23.	Wire mesh laid on canopies	27
24.	Recovery room prior to shield recovery	28
25.	Recovery room showing pinrail locations	28
26.	Shield advanced from shield line during recovery	29
27.	Advancing shields forming a movable bulkhead	29
28.	Location of stress meters and convergence and differential movement stations in panels 1 and 2	31

ILLUSTRATIONS--Continued

Page

29. Irad gage vibrating-wire stress meter..... 32
30. Hydraulic pressure gages and recorder on shield..... 32
31. Strata failure in terms of face advance..... 34
32. Map showing location of 196 monuments installed at the mine surface. 35

SAFETY ASPECTS OF LONGWALL MINING IN THE ILLINOIS COAL BASIN

by

Ernest A. Curth[1]

ABSTRACT

The Bureau of Mines and Old Ben Coal Co. concluded a cost-sharing contract in 1975, the objective of which was to demonstrate the Herrin No. 6 coalbed in southern Illinois can be mined safely by longwall methods. Under the agreement, three longwall panels are to be mined at Old Ben No. 24 mine.

A premining investigation laid the groundwork for specifying a roof-support system designed to control hazardous ground. Old Ben Coal Co. and Bureau engineers selected lemiscate-type roof shields. A rock mechanics program and geological mapping provide early warning capability and criteria for future equipment design. The effect of mining on the surface was monitored by a surface survey to develop subsidence-prediction criteria for the Illinois coal basin.

The first of the three longwall panels started up in September 1976 and was completed in May 1977. The major adverse condition was the occurrence of limestone concretions, called coal balls, in massive pods. The shields provided adequate roof control even in faulty ground. Operation of the second panel started on August 1, 1977, following panel-to-panel transfer of the equipment. The accident rate was low.

INTRODUCTION

The potential advantages of longwall mining over room-and-pillar systems include better ground control, generally easier compliance with safety standards, improved productivity, and higher resource recovery. Under the Bureau's Advanced Mining Technology Program, the Bureau and Old Ben Coal Co. concluded a cost-sharing agreement in April 1975, the objective of which was to demonstrate that the Herrin No. 6 coalbed can be mined safely by longwall methods.

[1]Mining engineer, Pittsburgh Mining and Safety Research Center, Bureau of Mines, Pittsburgh, Pa.

Unsuccessful attempts at longwall mining of the Herrin No. 6 coalbed were described by Moroni (5).[2] Six faces at Old Ben No. 21 mine failed between 1962 and 1971 owing to inadequate roof support by chocks. Experience at Orient Nos. 5 and 6 mines of Freeman Coal Co. was similar to that at Old Ben No. 21, and longwall mining was discontinued. If problems such as instability at the face and hazardous ground control are solved, longwall mining can be reconsidered as an alternative mining method.

Under the 1975 agreement between the Bureau and Old Ben Coal Co., three 460- by 1,740-foot panels were to be mined by longwall methods at Old Ben No. 24 mine, near Benton, Franklin County, Ill. The cooperative effort emphasized safety in design and operation, and included the following:

1. A premining study as a basis for equipment selection.

2. Design of a roof-support system to control hazardous ground.

3. Development of face equipment with safeguards.

4. Operation mode with safe face procedures.

5. A panel-to-panel transfer method aimed at the safe recovery, transportation, and reinstallation of equipment.

6. A ground-control and subsidence study of the effect of mining on rock strata and mine surface.

ACKNOWLEDGMENTS

The author is grateful for the cooperation of Old Ben Coal Co. and Dames and Moore, Geotechnical Consultants, in the preparation of this report. In particular, the author wishes to thank the following people, who generously contributed so much of their time by providing advice, documentations, and illustrations: E. T. Moroni, Senior Vice President at the Chicago, Ill., office of Old Ben Coal Co.; John Janes, Corporate Manager of Industrial Engineering, Gary Gray, Industrial Engineer, and Mark Cavinder, Longwall Coordinator, at the Benton, Ill., office of Old Ben Coal Co.; and P. Conroy, Ph.D., Associate of Dames and Moore, Chicago, Ill.

MINE DESIGN

At No. 24 mine, the Herrin No. 6 coalbed is intersected by four shafts at depths of 600 to 650 feet. Conveyor belts carry the utility-grade coal to the skip shaft. On the surface, the coal is processed through a Bradford breaker and is then conveyed to the surface storage and unit train-loading facility. In the mine, track is maintained for transportation of personnel and supplies. A surface structure, the Rend Lake Dam, is of concern in the prospective longwall-mining area.

[2]Underlined numbers in parentheses refer to items in the list of references at the end of this report.

The panel entries, in sets of three on 60- and 100-foot centers, are being developed by continuous-mining machines and are 14 feet wide except for the center entry, which is widened to 18 feet to accommodate the longwall power pack and the power center along the conveyor belt. The roof of the entries is supported by 5-foot resin bolts on a 5-foot grid. A sealing coat is applied to roof and ribs to protect them from the effects of the mine atmosphere (fig. 1).

The chain pillar on 60-foot centers is mined along with the longwall face, adding to its length. This practice is unique to longwall mining and designed to increase coal recovery. The three gate roads leading to the face are the headgate, the supply gate, which is tracked for transportation of personnel and supplies, and the tailgate.

The supply gate is designated as the intake escapeway; therefore, the trolley wire is not connected to the mainline and can be energized only by a switch at the inby end of the wire. Normally, the power is turned off; when the supply road is to be used, the vehicle operator must telephone the face attendant, who energizes the trolley wire. Whenever the trolley wire is energized, an alarm device, which is located at the parting leading to the longwall supply gate, flashes a light and sounds a bell.

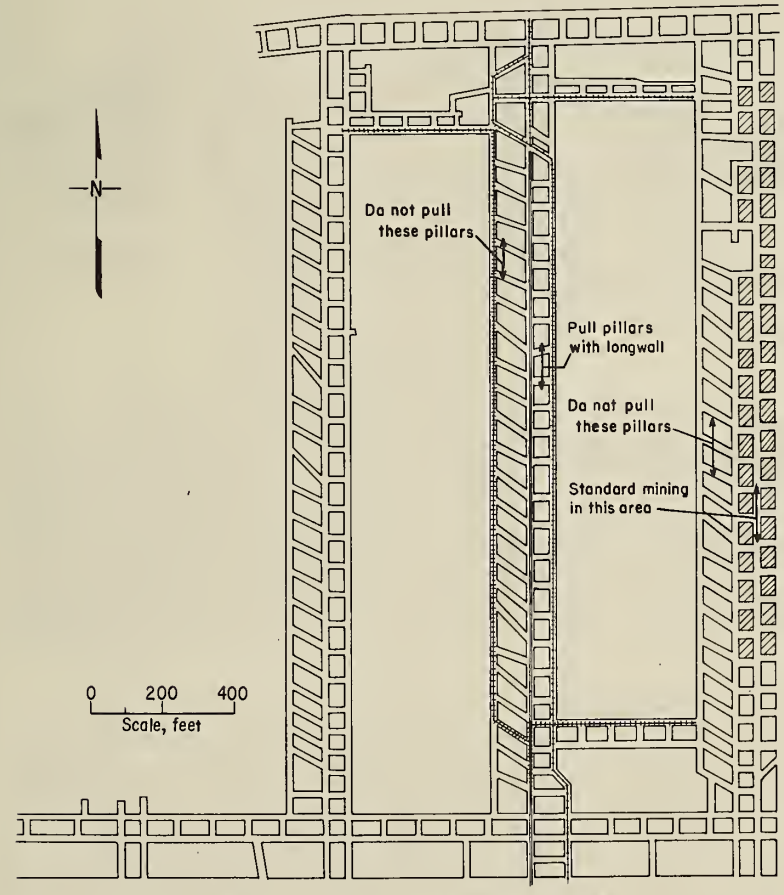


FIGURE 1. - Layout of panels 1 and 2.

Pillar extraction by continuous-mining machine from the panel adjacent to the tailgate preceded the longwall mining. Roof control at the tailgate may be affected by this preliminary extraction, because the pillar is only 86 feet wide and intersected by crosscuts.

PREMINING INVESTIGATIONS

Dames and Moore, Geotechnical Consultants, conducted the premining effort, under subcontract, to lay the groundwork for selecting a roof-support system designed to control hazardous ground. The following investigations were included:

1. Analysis of bearing capacity of roof and floor rock in the vicinity of the projected longwall area (2) (fig. 2).

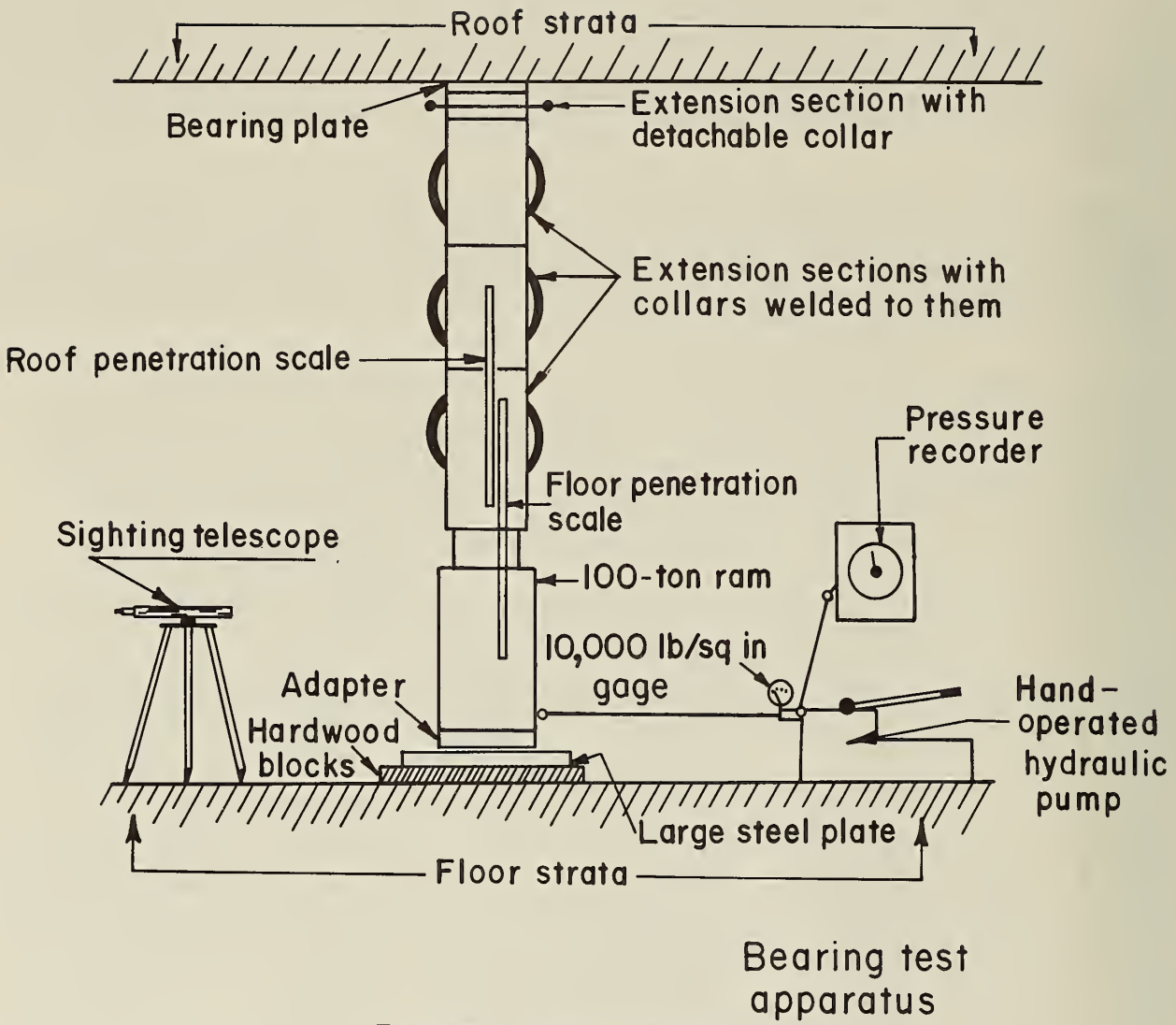


FIGURE 2. - Bearing test apparatus.

2. Geological study comprising rock-strata and faulting information.

3. Stress analysis on Rheinstahl chocks salvaged from a longwall face abandoned in 1967.

Dames and Moore concluded with the following:

1. A minimum of 6 inches of floor coal should remain intact. According to tests, the soft underclay fails at 300 lb/sq in when wet, whereas the floor coal can bear at least 1,000 lb/sq in. Therefore, the continuous mining machine in the panel development and the shearer at the face should cut to the roof rock and leave enough floor coal to maintain a mining height of 7 feet in the 8.5- to 9-foot-thick No. 6 coalbed.

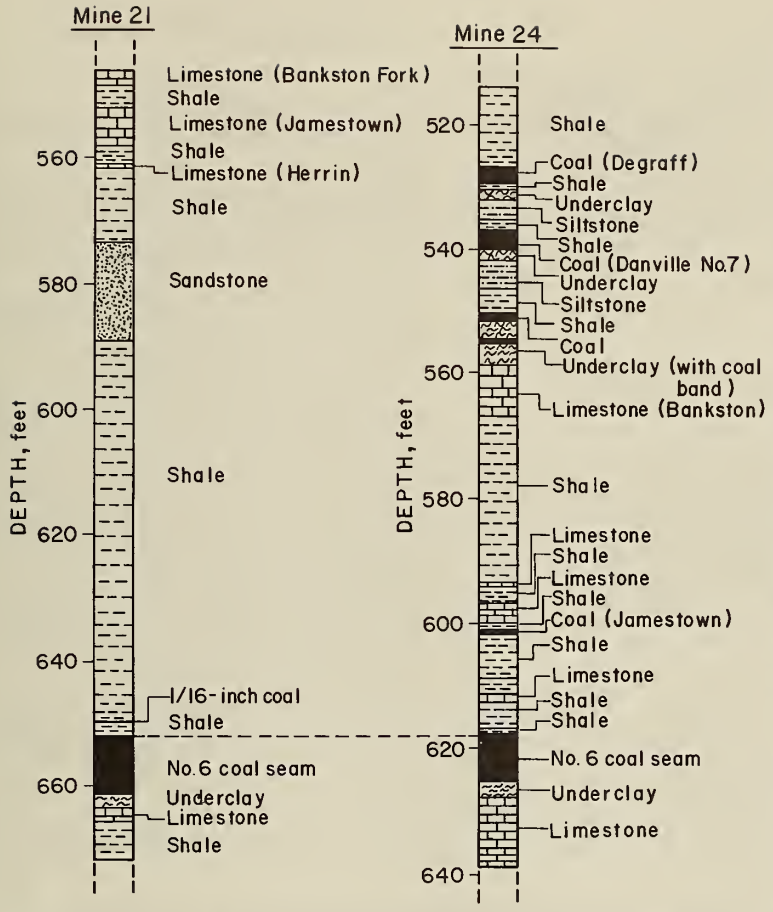


FIGURE 3. - Geologic columns according to cores from boreholes drilled from the surface of mines 21 and 24.

2. Considering strata separation and cantilever action in the very stiff strata, the roof-support system selected should have a mean load density of 9 tons/sq ft (1). Figure 3 shows the rock-strata columns according to cores from boreholes drilled from the surfaces of mines 21 and 24. Bureau personnel conducted geophysical logging in the NQ-sized borehole over longwall panel 1 of the No. 24 mine to determine physical properties of the in situ rock. In both mines, the immediate roof is formed by a very strong, fine-grained shale up to 70 feet thick that contains 20 to 30 percent siltstone. The main roof is composed of sandstone, limestone, and sandy shale. The floor consists of a soft underclay, 2 feet thick, underlain by limestone. A strata separation could occur between shale and limestone at 70 feet above the coalbed.

The direction of the main rock joints, N 80° E, roughly parallels the planned longwall faces and, therefore, favors caving. A large fault system, the Rend Lake Fault, was projected from earlier exposures. Trending north-south, it probably strikes within several hundred feet to the west of the proposed mining area and may affect the operation to some degree. The fault was found to be dry. Displacements of up to 50 feet occurred (fig. 4). A small spur fault, also trending north-south, occurs in panel 2 with a displacement of approximately 5 feet.

3. The roof-support system must provide a continuous overhead canopy and stability against lateral thrust.

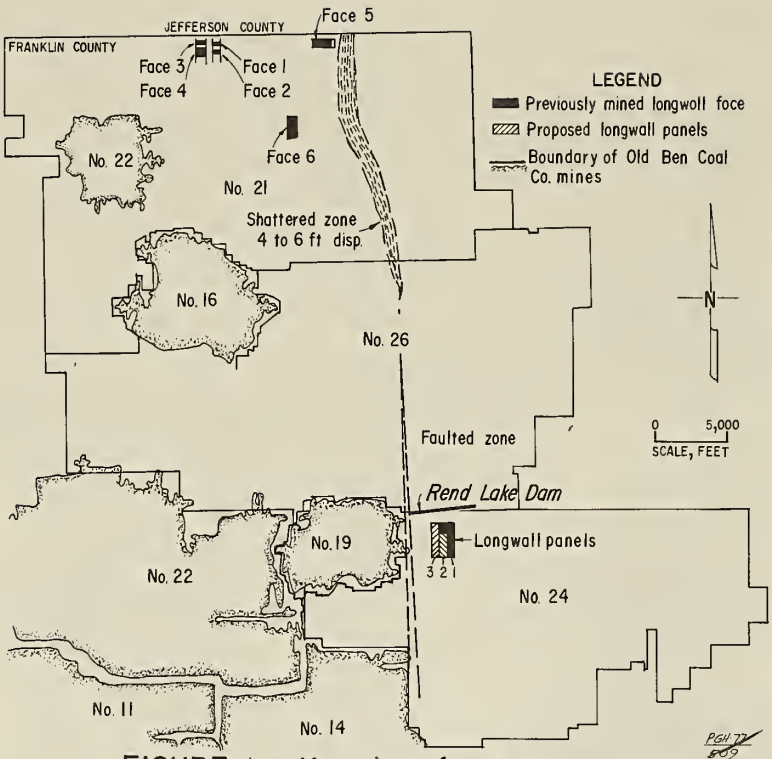


FIGURE 4. - Key plan of project area.

ROOF SUPPORT DESIGNED FOR HAZARDOUS GROUND

Recommendations resulting from the premining investigation led to the selection of roof shields. The primary advantage of the shields compared with other types of powered support is greater stability. It is a three-jointed, kinematically determinate system that can easily advance while the canopy is in contact with the roof. All other powered support systems are four-jointed, kinematically indeterminate, and require holding springs or other devices for stability. Other advantages of operation with powered shields are as follows (3):

1. Cavities do not have to be blocked by cribbing above the canopies.

2. The space where personnel work is fully sheltered by the shields from falling roof rock.

3. Manual cleanup work is not needed.

4. The shields advance without delay because the space is free of rubble and the structure is stable.

5. The "trampel" effect is less severe. (German miners use this expression to describe roof deterioration above powered roof supports caused by frequent setting and lowering.) Shields take fewer steps walking from gob to face, because shield canopies are shorter than chock canopies.

6. Control of weak strata can be achieved owing to the short span of exposed roof and the speed with which support is supplied.

Old Ben Coal Co. and Bureau engineers developed specifications for roof shields. The Thyssen[3] lemniscate-type roof shields RHS 18/30 L satisfied the following selection criteria for the safest possible roof supports:

[3]References to specific equipment does not imply endorsement by the Bureau of Mines.

1. A mean load density in excess of 9 tons/sq ft.

2. A hydraulic extension from 6 to 10 feet in one stroke without the use of extension members, so that cavities can be supported quickly.

3. A yield pressure of less than 6,000 lb/sq in, which provides a more reliable yield cartridge function than higher pressures.

4. A span of 12 inches between canopy tip and face before the shearer passes.

5. Each half of the base can be lifted individually to overcome obstacles or steps in the floor.

Figure 5 shows the shield during installation and figure 6 shows the side view of the shield in one-web-back position. Shearer and conveyor are also depicted to indicate relative position. The canopy tip of the lemniscate-type shield moves up and down in a nearly straight line, which is the center part of a lemniscate or figure-eight curve, and, therefore, the span between canopy tip and face remains approximately 1 foot.

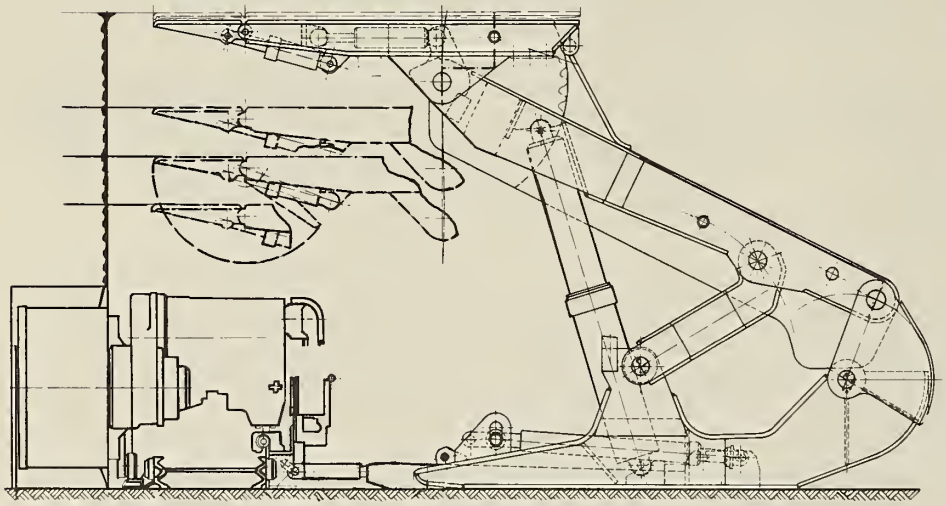
The shield weighs 17.5 tons and its overall length is 16.5 feet. The powerpack supplies a hydraulic working pressure of 4,350 lb/sq in; the shield is designed to yield at 5,500 lb/sq in, at which time spring-loaded relief cartridges open. Thus, the working pressure equals 80 percent of the yield pressure to maintain a strong thrust against the roof immediately after exposure and to prevent bed separation.

The shields are specially designed so that extreme power is concentrated in the legs. The two legs together can exert a thrust of 518 tons and yield a load of 656 tons; however, a mechanical disadvantage reduces the force that acts on the canopy hinge. At a 7-foot mining height, the shield can exert a thrust of 404 tons and sustain a load of 512 tons at yield.

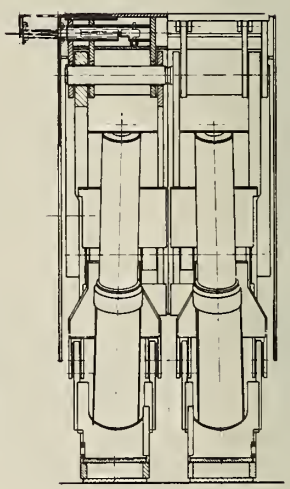
A minimum of 1 foot between canopy tip and face must be maintained to provide a safe clearance between the rotating drums and the canopy tip. The shearer drum is 2 feet in width so that, after the shearer has cut by, the roof exposure is widened to 3 feet. This span is reduced to 1 foot again after the shields are advanced 2 feet. However, the entire shield can be brought 2 feet closer to the face and operated in the none-web-back or conventional mode by lowering the canopy extensions, which are 2 feet long. In the closed position, the shield is rated to yield at a mean load density of 12.5 tons/sq ft before the shearer passes and 10 tons/sq ft after the shearer has cut by.

The canopy is designed to maintain a stable roof contact during the shield advance by the action of a ram which swings it against the roof and by having a ratio of front portion of canopy to rear portion of not greater than 2. The canopy tip load exceeds 5 tons at yield.

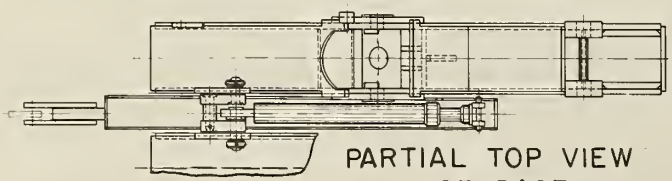
FIGURE 5. - Shield during installation.



SIDE VIEW



FRONT VIEW



PARTIAL TOP VIEW
OF BASE

FIGURE 6. - Thyssen RHS 18/30 L roof shield.

The base, including the weight of the shield, exerts a floor pressure of 311 lb/sq in at yield. The advancing shield can overcome uneven or soft floor or obstacles because each half of the base can be lifted individually by its double-acting leg. The base is self-cleaning in that debris and slack coal can pass into the gob through a sufficient number of open spaces in the base while shields are advanced.

The tendency of the shield base to dig into the floor is counteracted by having the double-acting ram mounted in an inclined position. The ram acts through reverse linkage to raise the base slightly and to exert the full force of the piston end when the shield is pulled up to the conveyor. Also, the shield is designed so that the forward edge of the base is ahead of the projection of the canopy hinge joint.

Indication of marking pin	Nominal prop load		
	40 Mp	60 Mp	100 Mp
	Pressure indication in Mp		
	0	0	0
	10	15	22.5
	20	30	45
	30	45	67.5
	40	60	90

1 Mp = 1.1 short tons

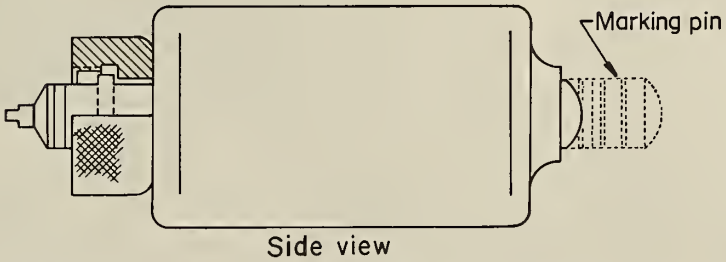


FIGURE 7. - Piston-type load indicator with load chart. (Megapond (Mp) is a unit of measure used in the Federal Republic of Germany.)

Each shield is controlled from the adjacent unit to protect the shield operator. The shield can be advanced while automatically maintaining a soft roof contact to keep the roof intact after exposure. But a manual override control allows the legs to retract to overcome roof irregularities. Evidently, maintaining roof contact by automatic control is preferable to lowering the canopies manually.

The hydraulic system must have sufficient capacity to coordinate the speed of the shield advance with the shearer haulage speed of 24 ft/min, and to minimize the delay of supporting the roof after exposure. Hydraulic pressures in the legs are a measure of roof loads. Therefore, each leg is fitted with a piston-type load indicator to monitor roof loads and to provide early warning of approach to the yield point (fig. 7).

A safe travelway along the face is provided.

Dust originating from the gob during shield advance is controlled by automatic sprays mounted on the canopies. The working space is sealed by sideplates fitted between the shields and maintained by springs and hydraulic cylinders.

A permissible lighting system was ordered and a group of luminaires was tested at the face, but the U.S. Mining Enforcement and Safety Administration (MESA) suspended permissibility of the luminaires pending further investigation.

To assure the safe performance and reliability of the shields and to discover flaws in design, components and entire prototypes were subjected to a battery of tests at the German Research Center for Rock Mechanics and Roof Support in Essen. Components also underwent the tests mandatory for application in German mines at the appropriate center (fig. 8).

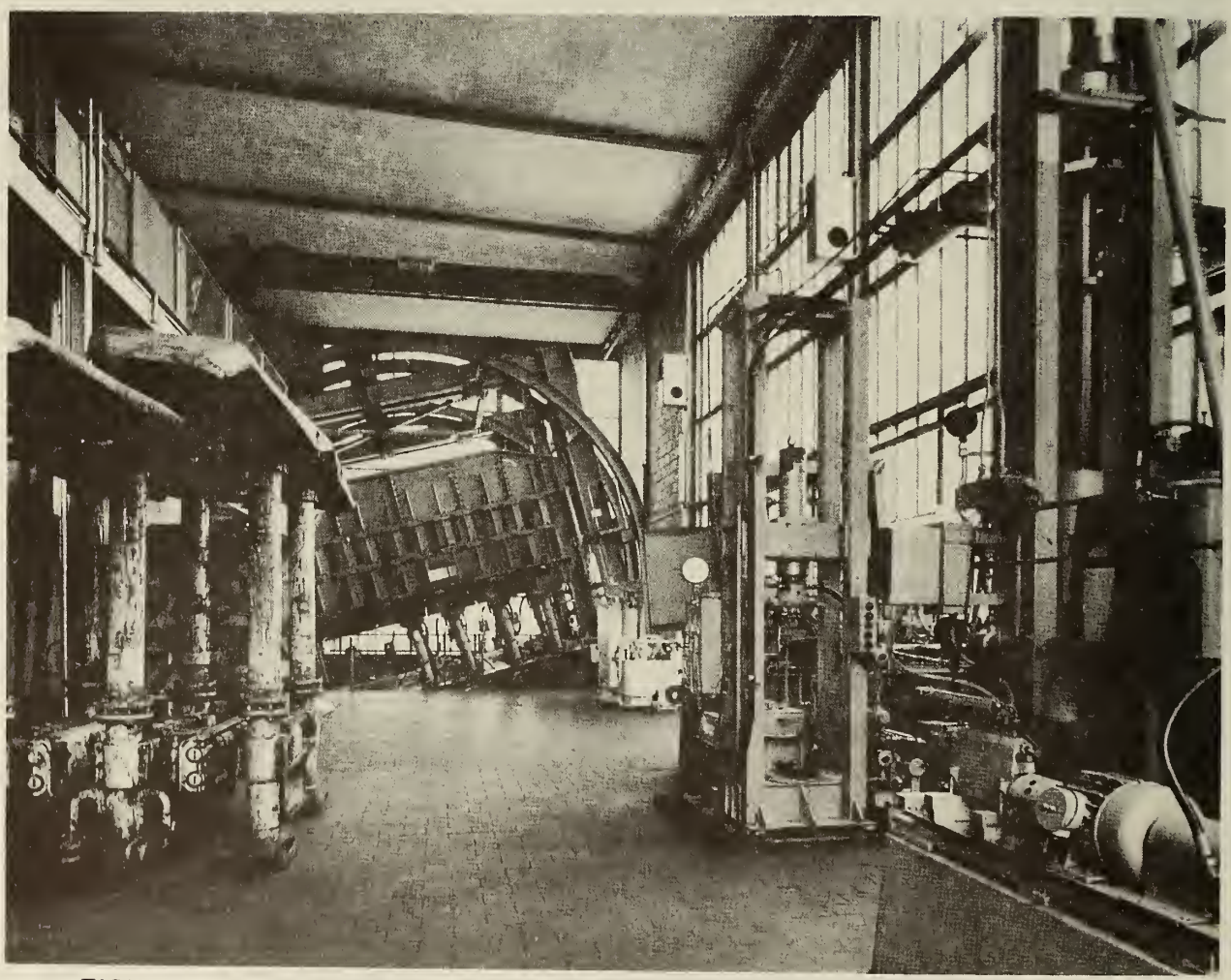


FIGURE 8. - Roof support testing laboratory, test frame for powered supports in the background.

The intensive testing program at the Essen laboratory included the following investigations (4): The forces exerted on roof and floor, the support resistance, and the bending moments in the floor skids; the penetration of the floor skids into the floor; the strength of the shield at one to three loading cycles; the durability of the shield under pulsating load; the function and operation mode of the canopy; the canopy cylinder and the angular positioning of the canopy; the stability of the roof support; the protection against flushing; the width and height of the travelway; behavior of the shield on uneven floor; and the sideways mobility of the double-acting ram.

FACE EQUIPMENT WITH SAFEGUARDS

The following roof support equipment was procured:

1. A shield line of 95 units on 59-inch (1.5-meter) centers to accommodate the conveyor pans, which are 59 inches (1.5 meters) long, so that the middle of the pan can be attached to the double-acting ram, thereby minimizing the stress on the pan connections due to eccentric forces. Three of the shields are placed across the headgate and two in the tailgate.

2. A powerpack consisting of two high-pressure pumps, each capable of operating the full shield line by supplying 25 gal/min to the dual pressure system; one mixing tank to prepare the 5-percent soluble-oil-in water emulsion that powers the system; and two filters.

3. 150 hydraulic single props with a range of 7 to 10 feet. Connecting them to the hydraulic pressure system via hose and quick-connect, pistol-type coupling will extend them to exert a thrust of 30 tons, if properly blocked. Each prop yields at 44 tons. Bleeding the fluid retracts the prop.

The hydraulic props are placed into the three gate roads and crosscuts less than 80 feet from the face and remain there until the face approaches within 3 to 5 feet (fig. 9). According to European and U.S. studies (3), rows of single props, by virtue of early-bearing capability, control convergence in gate roads better than late-bearing cribbing, which relies on rock deformation for building up the support strength (fig. 10). Several props in the supply gate, properly blocked and equipped with hydraulic gages, provided an early-warning system against excessive abutment pressures, which never occurred (fig. 11).

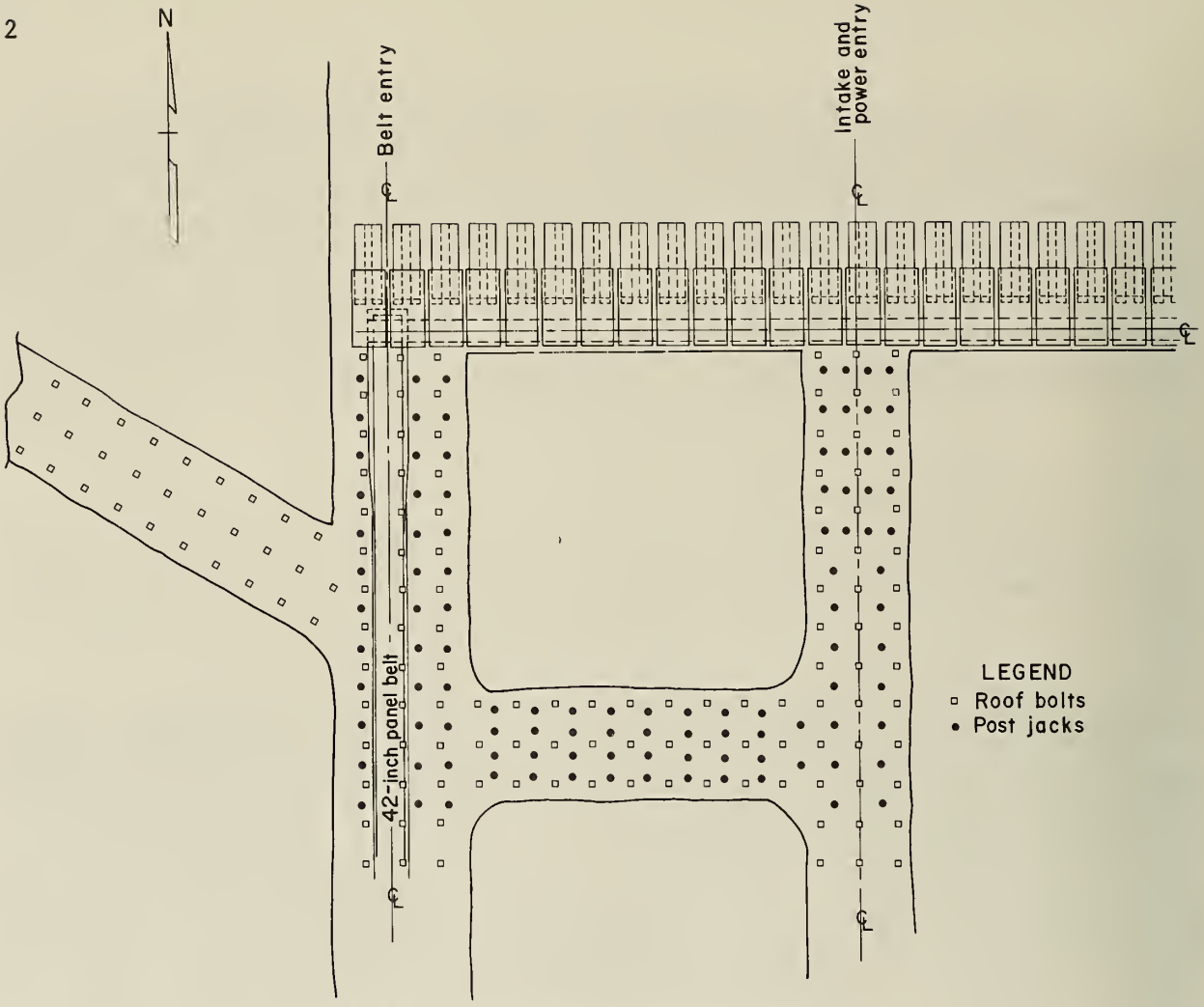


FIGURE 9. - Roof support plan for gate roads.

The following mining and transportation equipment is in use:

1. A modified Eickhoff EDW 2 X 100 L double-ranging drum shearer with two air-cooled, 100-kilowatt motors; 54- by 24-inch drums with two-start spiral vanes turning at 63 rev/min; 4-inch picks, 26 on the vanes and 24 on the disks; each drum equipped with a cowl to clean the shearer track; dust control by water sprays integrated with the drums and mounted on the machine frame; and a 26-millimeter haulage chain with hydraulic tensioners mounted on the head and tail terminals of the conveyor and connected with the hydraulic-pressure system of the shields. The tensioners compensate for the elastic stretch of the haulage chain, which may amount to as much as 2.1 feet for a 470-foot face and a 26-millimeter chain. They provide uniform chain loading regardless of the position of the shearer along the face. The shearer is safely trapped by guide tubes on the gob side; on the face side, the shearer is guided by rollers that run on the L-shaped ramp plates.

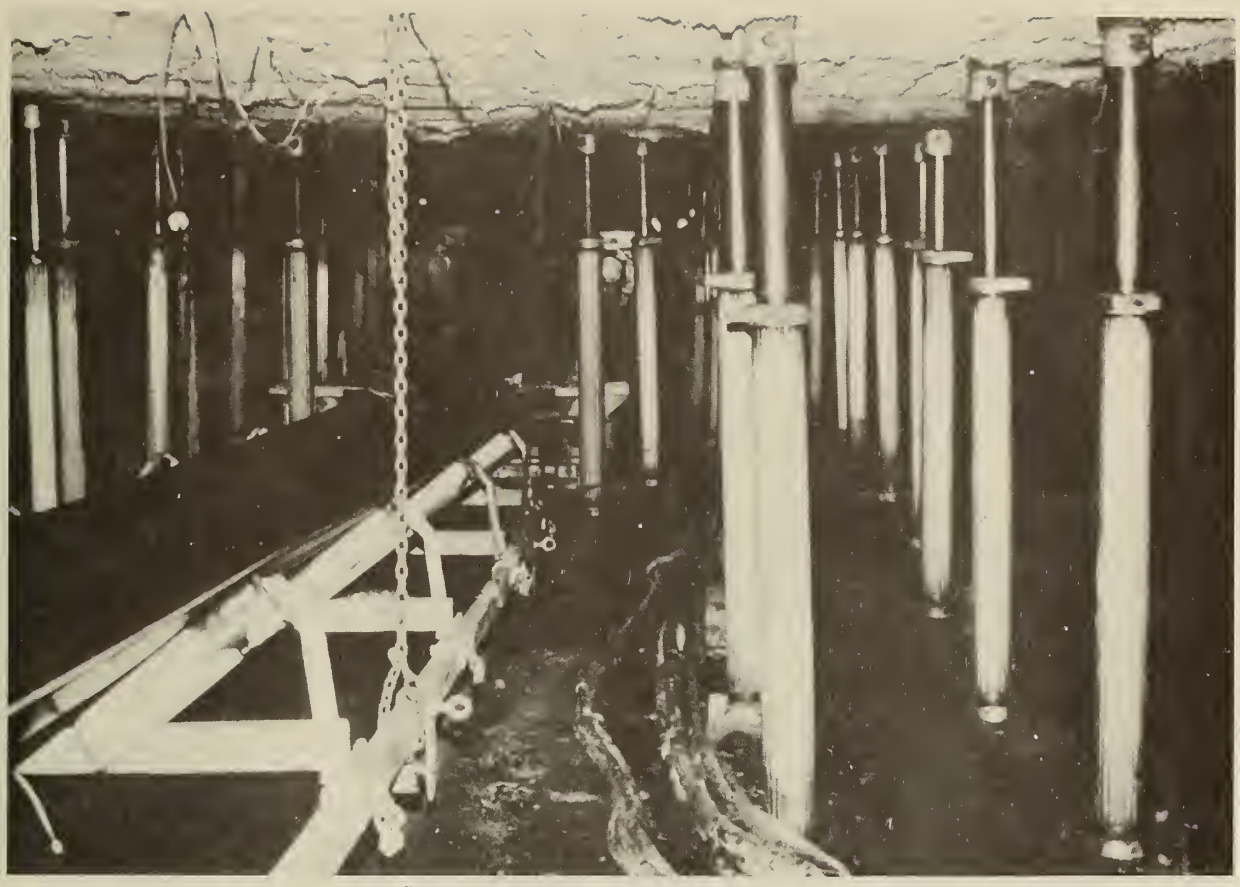


FIGURE 10. - Props in gate road.

2. An Eickhoff EKF 3 single-chain conveyor with one 150-horsepower drive at each end. The drives are on the gob side and parallel to the conveyor. Controls for the belt conveyor, stage loader, and face conveyor are interlocked in sequence so that belt stoppage will shut down stage loader and face conveyor. But the face conveyor can be restarted only by activating a reset switch located at the stage loader after a warning is sounded over the pager-phone system. Pushbutton emergency cutoff stations are placed 50 feet apart along the pan line. If the conveyor is shut off at such a station, it must be restarted at the same station. Before the conveyor is restarted by activating the switch, a warning must be sounded over the pager-phone.

3. A set of accessories, including spill plates, ramp plates, and a Bretby cable handler, which protects the shearer cable and water hose from mechanical damage.

4. A stage loader mounted on four rubber tires, attached to the face conveyor head, and swiveled on the belt tailpiece, which allows retracting of the conveyor belt in 12-foot increments. A chunk breaker is suspended so that material that cannot be crushed can pass.

FIGURE 11. - Load monitoring.

5. A 42-inch retractable belt conveyor with solid structure suspended from the roof by chains.

6. A power center for 440 volts.

7. A communication system of pager-phones which are placed at 100-foot intervals along the shield line.

FACE INSTALLATION AND PERSONNEL TRAINING

The face equipment was assembled in a 21-foot-wide staging room that was tracked with three rails to facilitate moving the shields into place. A 12-foot-high rigging chamber was excavated at the head of the staging room and equipped with two overhead cranes and an auxiliary hydraulic power supply. The face conveyor was installed first, with its chain, ramp plates, spill plates, and the cable handler. The shields were lowered into the mine and hauled on flat cars to the rigging chamber in four parts, as follows: each leg, with its half of the base and the gob shield; the canopy; and the double-acting ram. Each shield was assembled in the rigging chamber on the track and maneuvered into place in the shield line by battery-powered front-end loader. The shearer arrived unassembled and was taken to the underground shop. After assembly, it was hauled to the staging room and introduced over the head end of the pan line by use of a front-end loader (fig. 12). Head drive and stage loader were installed at the face. After the entire electric system was inspected and approved by MESA, the face operation started on September 3, 1976.

Prior to initiating the face operation, Old Ben Coal Co. conducted an intensive longwall training program for face crews, United Mine Workers of America officials, supervisors, and maintenance personnel. Management staff, including Vice-President--Operations and Corporate Director of Safety, supervised the introductory training at nearby Rend Lake College. They explained

FIGURE 12. - Installation of Eickhoff shearer.

the reasons behind Old Ben Coal Co.'s decision to reintroduce longwall mining. Sequence of operations was demonstrated on a three-dimensional model of the longwall section (fig. 13) and hazards peculiar to longwall mining were identified, such as whipping shearer-haulage chain, flying particles of coal and rock, and bursting hydraulic hoses and fittings.

Longwall section crews consisted of two shearer operators, five shield men, one mechanic, and one foreman.

At the face, manufacturer's service personnel and industrial engineering staff provided hands-on training to each crew member for proper operation of each piece of equipment. Actual production trial runs were made, and machinery was operated under one-on-one supervision.

Maintenance personnel spent 2 days at the Old Ben Coal Co. electric shop for training in trouble-shooting techniques after intensive study of the wire diagrams. Two more days, at the Rend Lake College vocational facility, were devoted to instruction in operation, maintenance, lubrication, and electric and hydraulic circuits of the shearer. The shearer manufacturer sent an instructor and supplied a cutting motor and a haulage box wired for hands-on training.

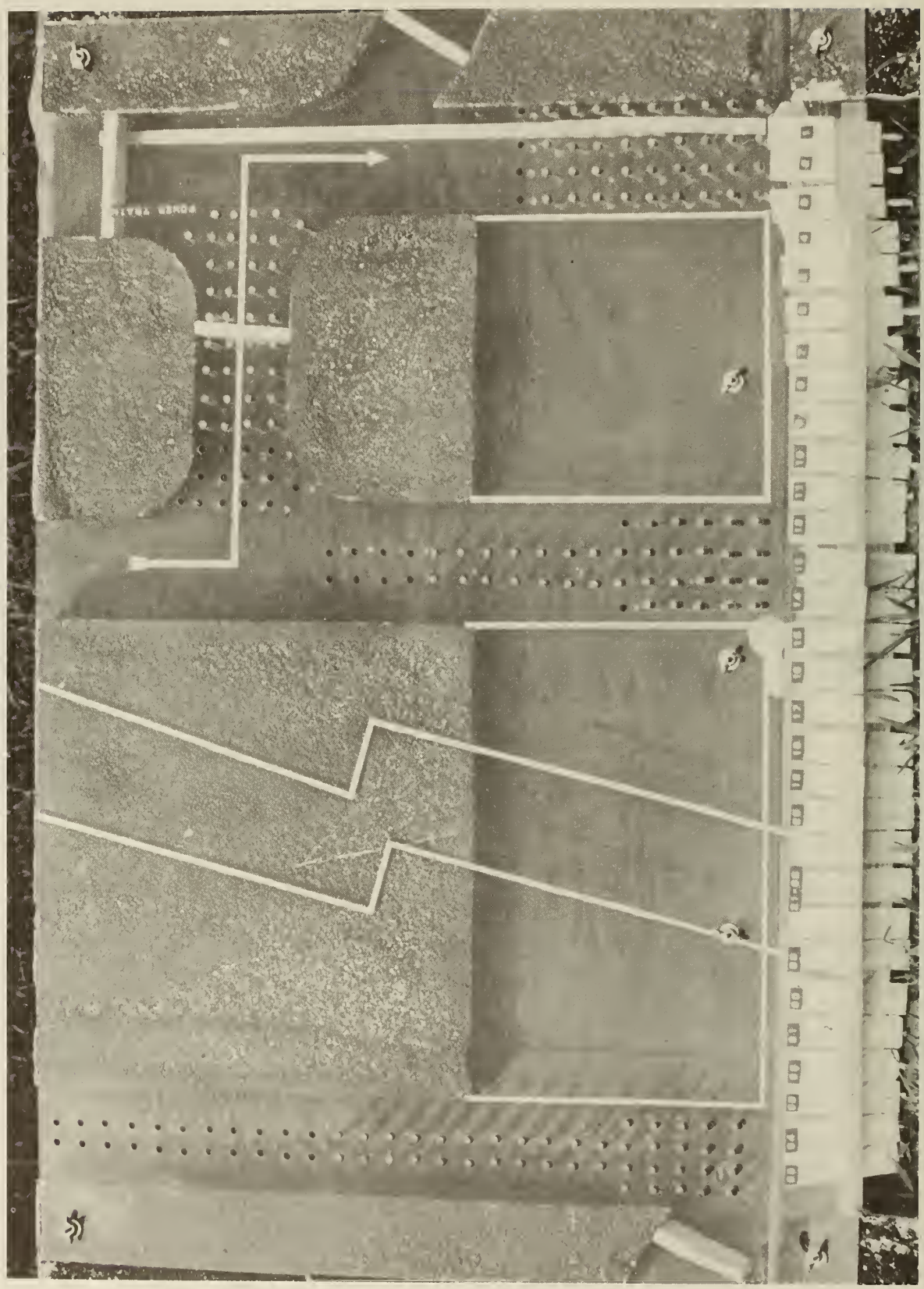


FIGURE 13. - Three-dimensional model of face operation.

Safety and job skills training and retraining have been conducted continuously since the start of the operation.

FACE OPERATION

The operation mode is unidirectional. Crews remain in intake air while the shearer is cutting and the shearer cleans its track before the conveyor is rammed over the face. The intake air, at a volume of 25,000 cu ft/min, reaches the face via the supply gate, from which the main split of 20,000 cu ft/min travels to the tailgate; the balance, 5,000 cu ft/min, goes to the headgate. These air currents split again, each forming two secondary currents. One of each of these secondary currents reaches the bleeder via the gob; one of the remaining currents returns to the tailgate, and the other returns to the headgate panel return-air course. The sensor of the methane monitor is mounted on the conveyor taildrive. The power is cut off if methane concentration reaches 1 percent. Figure 14 indicates the mode of operation as follows:

1. Start.....	The shearer at tailgate.
2. Flitting--tailgate to supply gate.	The shearer cleans its track and the conveyor is rammed over to the face after the shearer has flitted by.
3. Cutting--supply gate to headgate.	The crews remain in the intake air. The shields are advanced from the one-web-back position toward the face after the shearer has cut by.
4. Flitting--headgate to supply gate.	The shearer cleans its track and the conveyor is rammed over (fig. 15).
5. Cutting--supply gate to tailgate.	The crews remain in the intake air. The shields are advanced from the one-web-back position toward the face after the shearer has cut by.

The unidirectional mode, as described, allows the tail-side drum to do most of the cutting. This is an advantage because the tail-side cutting motor exclusively drives the drum, whereas the head-side cutting motor also powers the hydraulic pumps for the shearer haulage and the actuation of the ranging arms. A bidirectional operation mode would offer only a small gain in production because the turnaround maneuver at the face ends requires more time and would be even slower without hydraulic actuation of cowls. (The system of hydraulic cowl operation, which came with the shearer, failed, and the panel was extracted with manual cowl operation.)

Roof safety in longwall mining depends on adequate load-bearing capacity, stability, and reliable function of the roof support. The characteristics of shields, such as the ability to advance under soft roof contact, the speed of advance, great thrust against the roof when set, and a load density at yield

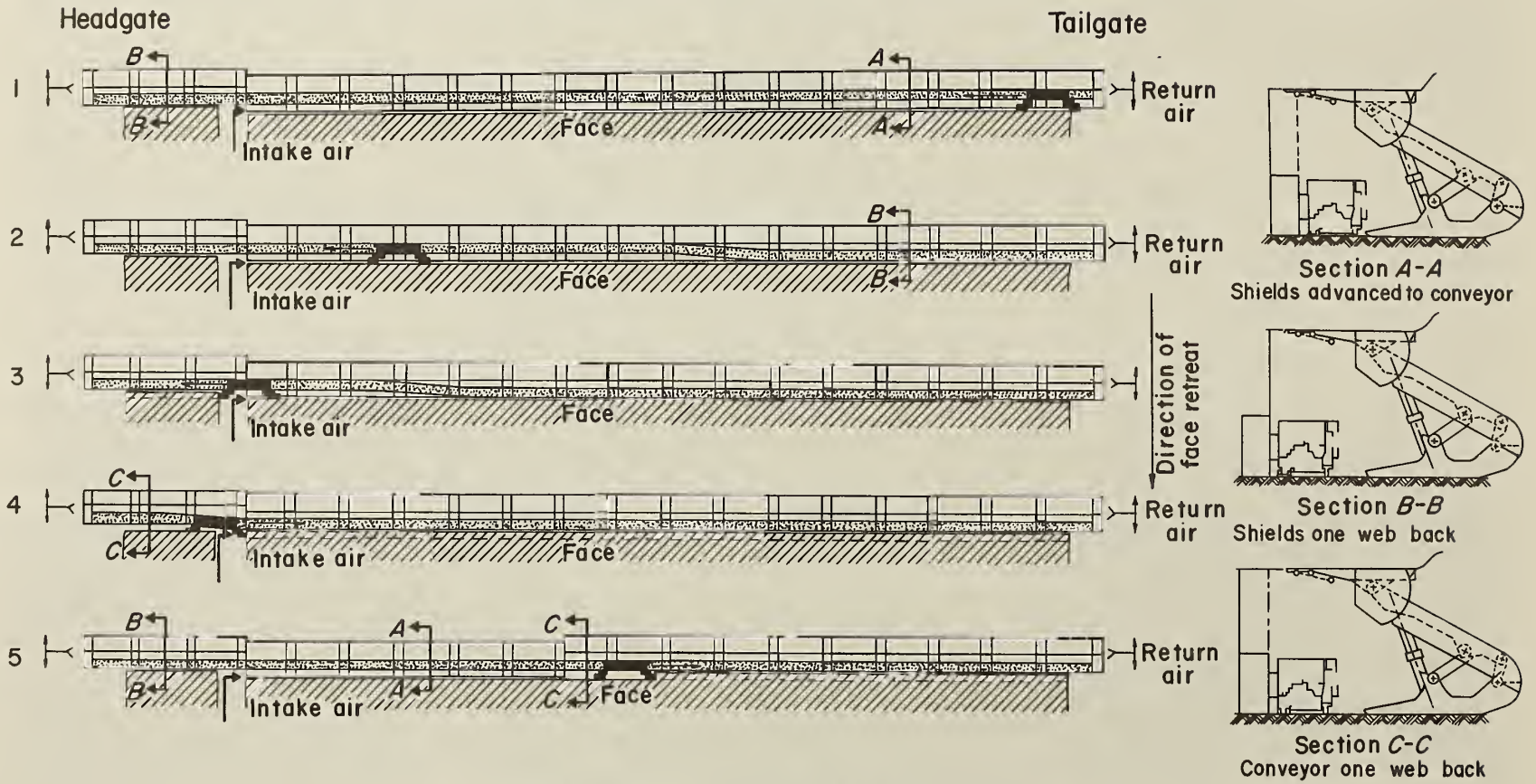


FIGURE 14. - Longwall mode of operation.

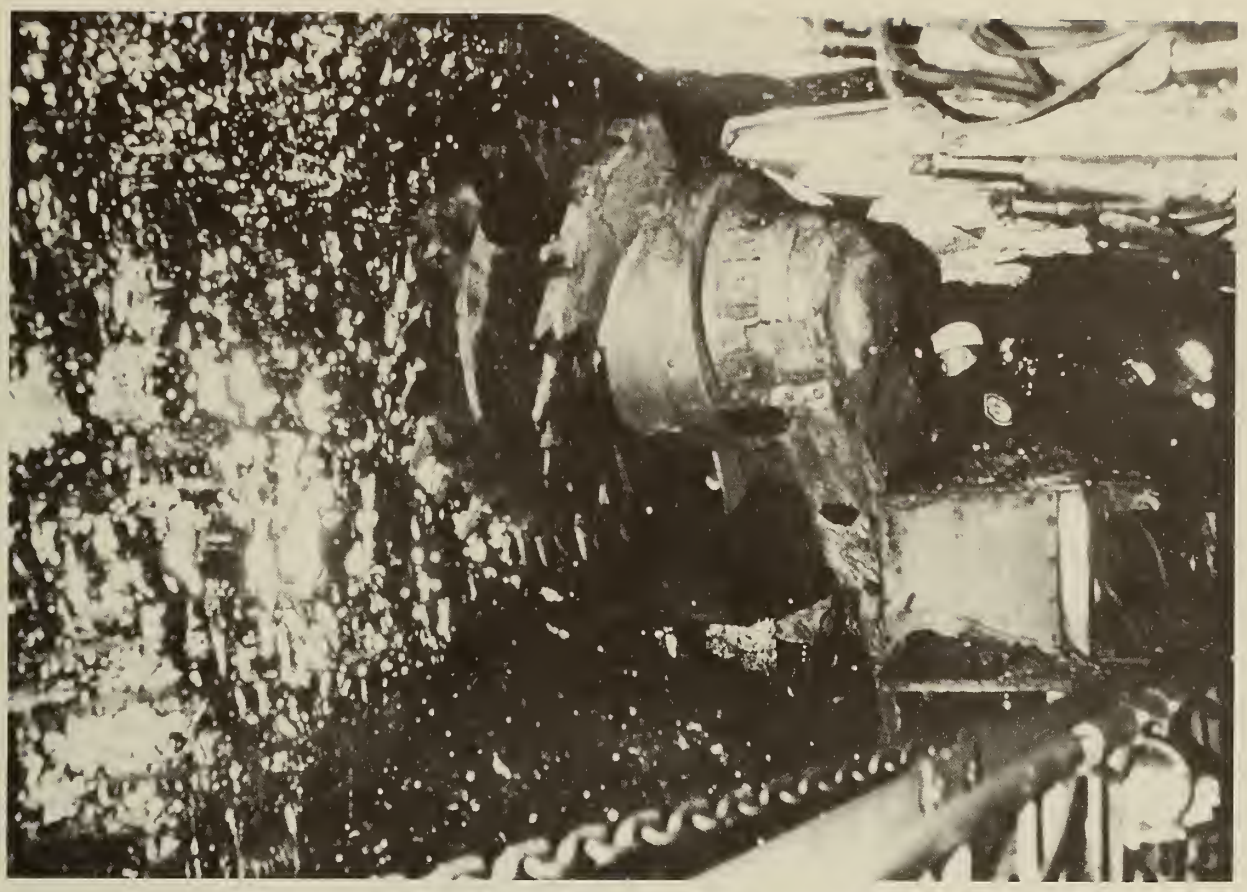


FIGURE 15. - Shearer at headgate.

up to 12.5 tons/sq ft contributed to roof control at the face (figs. 16-17). The shields maintained roof control even when the face struck a roll that displaced the entire coalbed and caused rock falls of 20 feet, well above the maximum shield height of 10 feet.

The hydraulic system must be inspected and adjusted periodically. Holding the setting pressure for 1 minute at each shield brings the full pump pressure to bear. The reliable function of the yield cartridges can be tested by a pump-with-gage unit, which is provided by the manufacturer. Moisture around the valve block indicates that the shield is yielding. Only a few shields were yielding during the extraction of the first panel; however, at the completion of the panel, when the face cut through to the recovery room on May 11, 1977, approximately 75 shields in the midsection and tail end of the face yielded until they were advanced to the bolted roof of the recovery room.

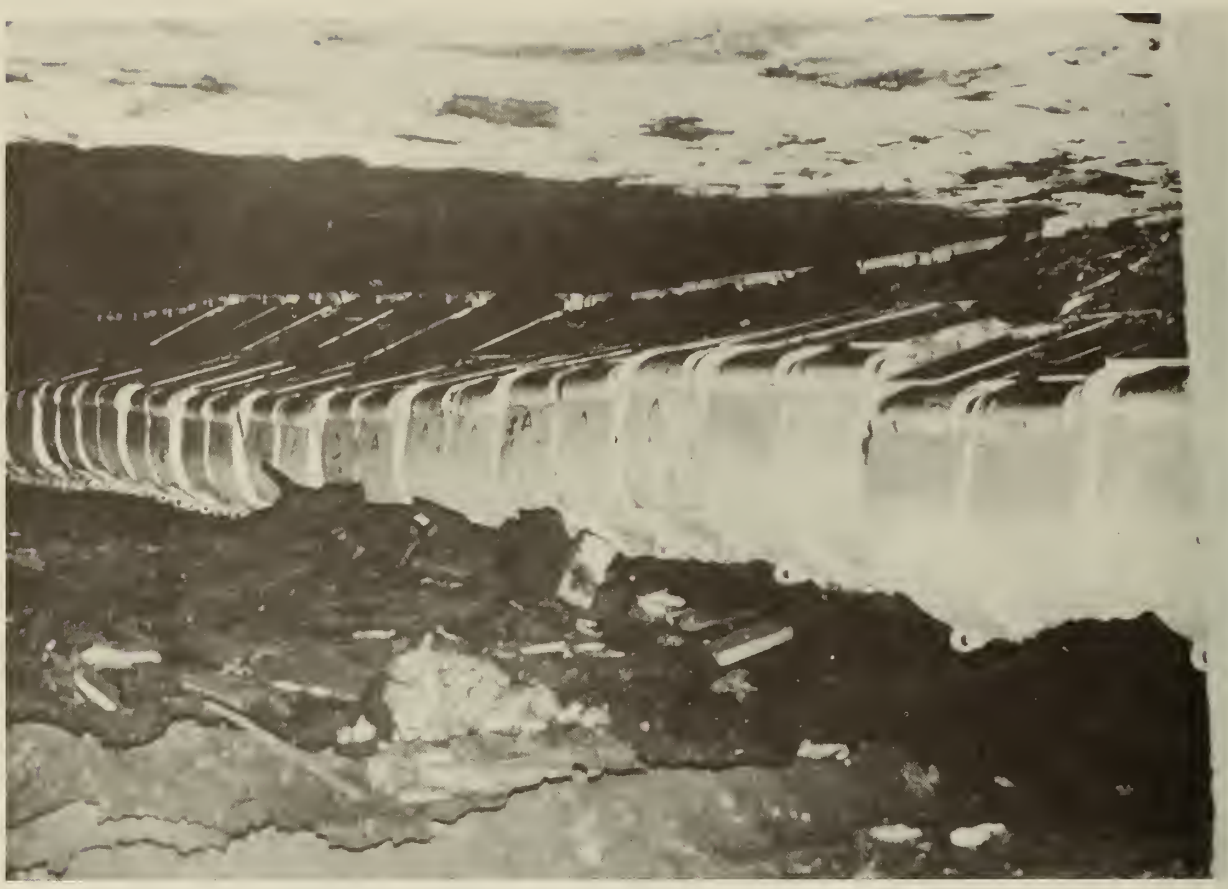


FIGURE 16. - Gob before the first roof fall.

Maintenance of shields was generally satisfactory. The most frequent defect observed during operation and involving all units was the breaking of shearpins in the shield-advancing arrangement, which is a crank-type gear that allows each skid to be lifted individually during the shield advance to obtain a level difference of approximately 19 inches measured at the tips of the skids. The shearpin, which secures a ring-type spacer, cannot sustain the stress resulting from the operation of the advancing ram and breaks. Both the manufacturer and Old Ben Coal Co. have developed versions of the shield-advancing gear that are more reliable, but reduce the floor-level difference that the shields can negotiate. Other maintenance problems reported were deformation of a few side plates and cylinders and of some canopy tips.

Pillar extraction in the panel adjacent to the tailgate by continuous-mining machine did not noticeably affect the longwall face, despite the fact that the coal pillar between the longwall and pillar extraction operations was only 86 feet wide and was intersected by crosscuts. The tailgate came under some roof pressure, which was sustained by the hydraulic jacks, the two shields in the tailgate, and cribs built across the crosscuts.

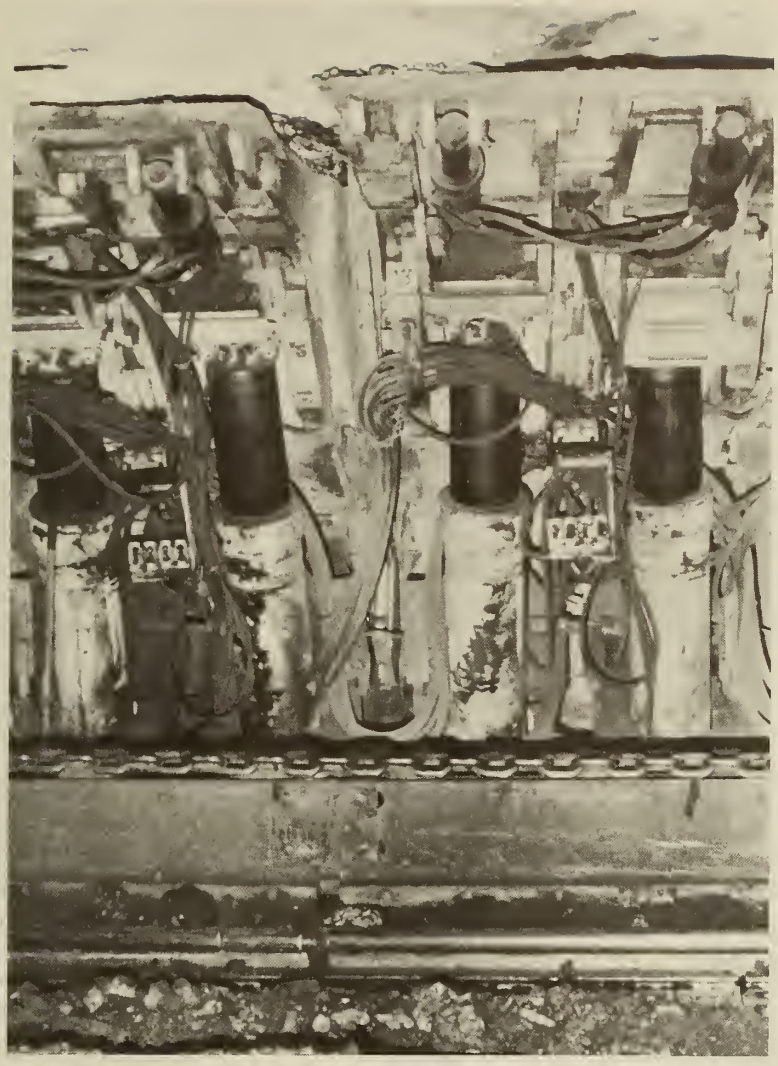


FIGURE 17. - Shield contact with roof.

The occurrence of sideritic limestone concretions called coal balls, which ranged in size from golf balls to footballs, posed the greatest problem. According to the Illinois State Geological Survey, coal balls occur in a black marine roof shale at the boundary with a grey shale, which covers most of the area of the Herrin No. 6 coalbed. At first, the tailgate development struck two large pods. The retreating longwall face encountered solid accumulations at the tail end and in the center (fig. 18). Unlike pillar extraction, a longwall cannot bypass such problem areas. It took 7-1/2 weeks to work through the pods, which had to be blasted in places where they displaced the entire coalbed or stuck to the floor. A large number of shearer picks was consumed and numerous bit blocks on the shearer drums were cracked. The bit blocks were temporarily welded until a new tail-side drum could be procured. The Bradford breaker on the surface was damaged, as were the stage loader and chunk breaker underground.

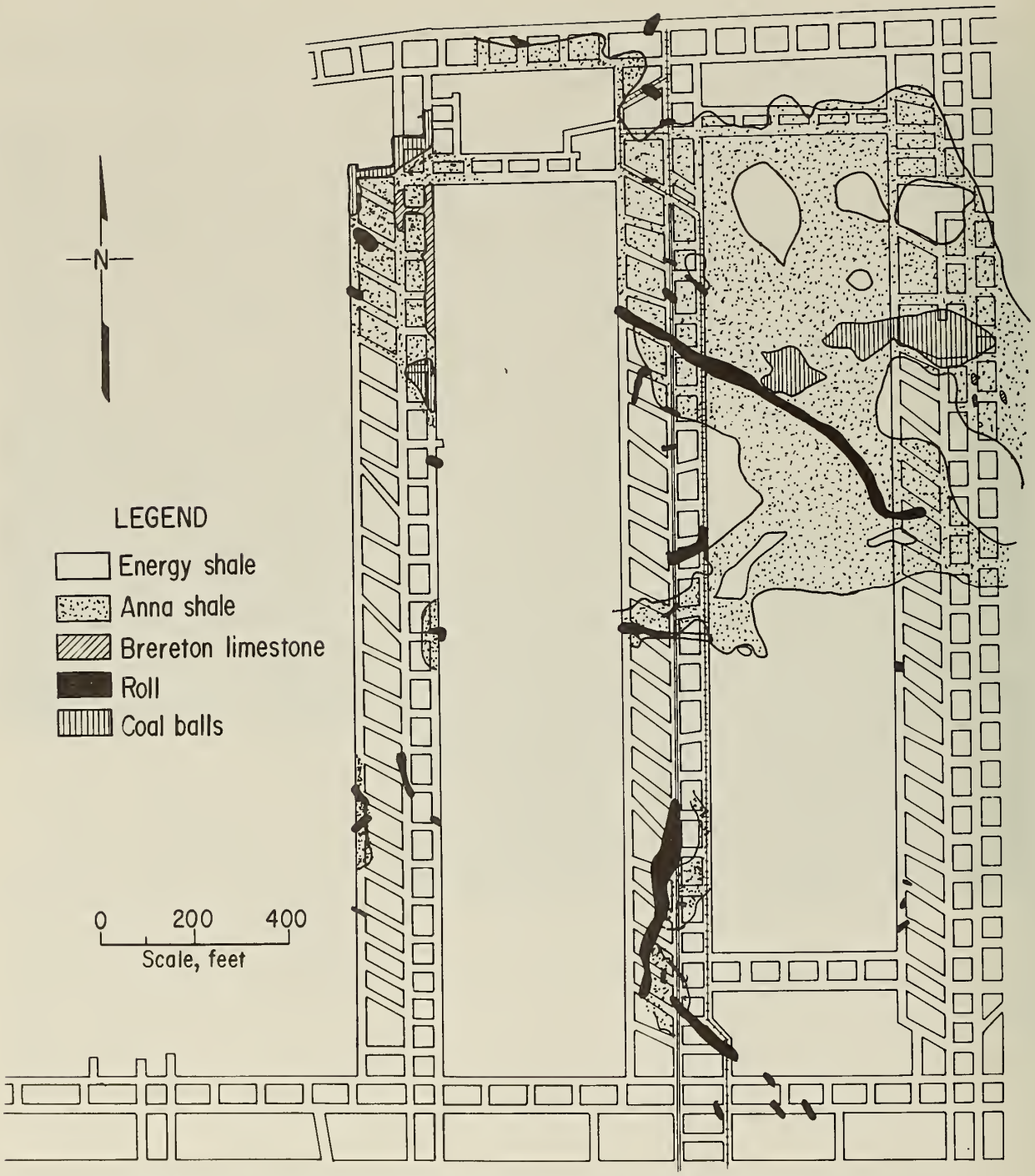


FIGURE 18. - Map of panels 1 and 2 by Illinois State Geological Survey.

The first panel yielded a total of 205,000 tons of coal from a length of 1,734 feet during 162 working days (fig. 19).

The following delays, expressed as percentage of the total available working time, occurred: Shearer downtime, 17 percent; other longwall delays,

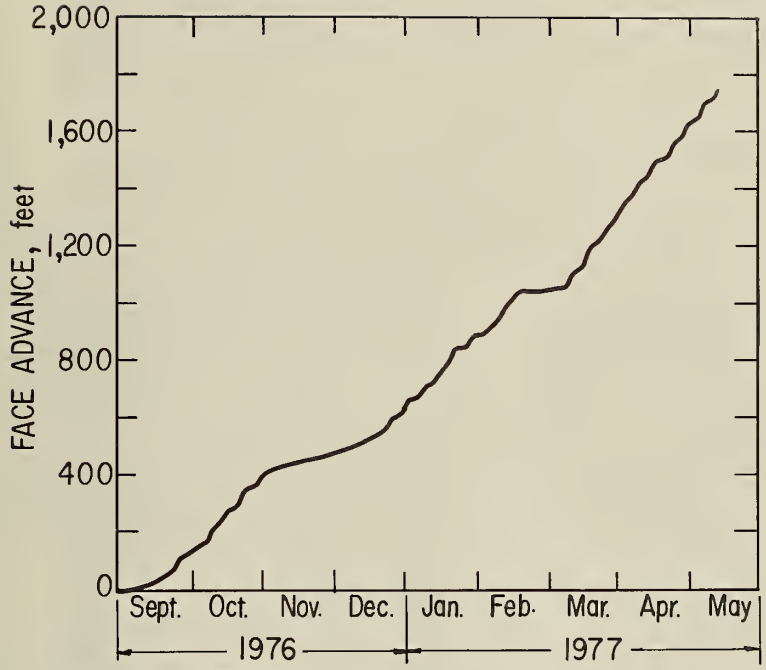
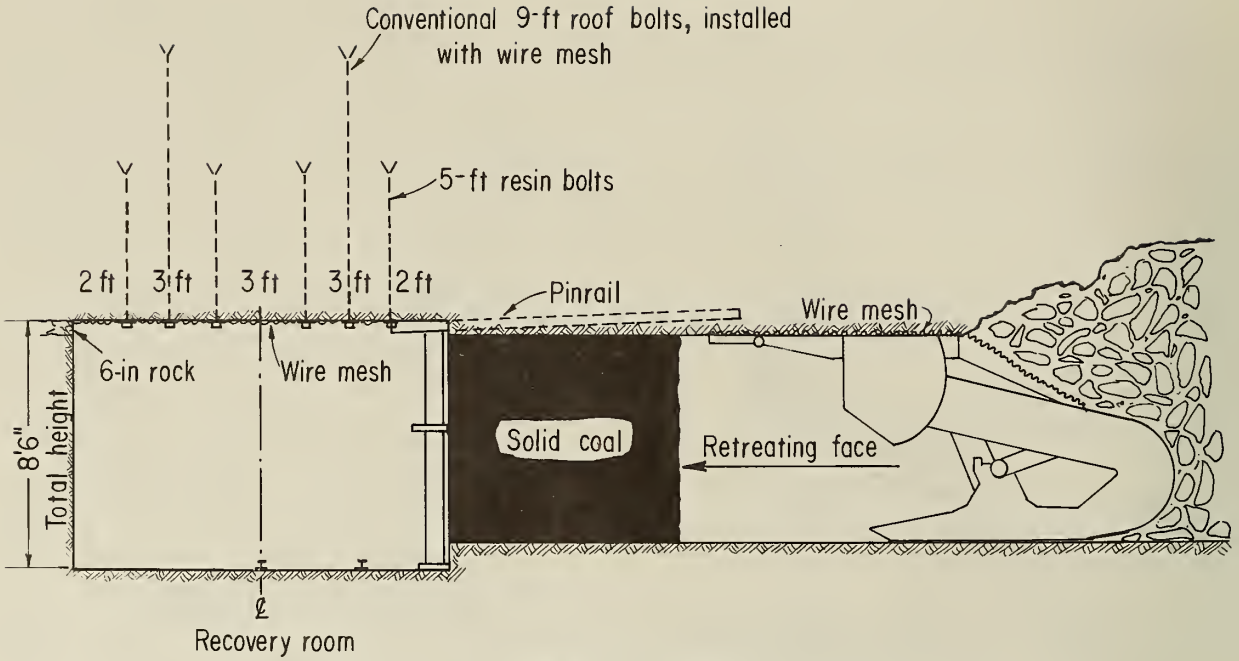


FIGURE 19. - Face advance in terms of time, measurements taken in supply gate.

including roof control, 13 percent; nonlongwall or external delays, 11 percent; conveyor downtime, 6 percent; and stage loader downtime, 4 percent. The sum of the delays is 51 percent; therefore, the shearer was operating 49 percent of the available working time. The cutting web averaged 21.4 inches in depth with the 24-inch wide drums and the mean quantity per cut was 211 tons of coal. Daily averages in April were 8.2 cuts, 14.5 feet of advance, and 1,708 tons of coal. At peak performance, the shearer achieved 15-1/2 cuts, producing 3,317 tons of coal in 24 hours.

PANEL-TO-PANEL TRANSFER

Recovery of longwall roof support equipment is always onerous and requires circumspect planning and execution, but shield recovery poses an unusual problem because part of the structure is under the caved rock. The weight and size of the shield (35,000 pounds, 6 feet high, 5 feet wide, and 16.5 feet long) added to the difficulty of moving. The unique method of recovery, designed to handle the units in the safest possible manner, is outlined as follows: The longwall face holed through into a recovery room, which had been prepared to receive the face equipment at the completion of the panel. A height of 8 feet 6 inches was provided by cutting into the roof rock. The roof was supported by 5-foot resin bolts and 9-foot conventional bolts on a 3-foot grid combined with No. 9 gage chain link fence-type wire mesh (fig. 20). Forty-pound rails, 15 feet in length, were installed in the roof shale to extend over the longwall block. The inby ends of these rails came to rest on the shield canopies when the shields approached the face prior to holing through. The outby ends were



VIEW LOOKING WEST

FIGURE 20. - Recovery room prior to holing through.

supported by hydraulic props set in the recovery room. Track was laid to facilitate removal of the shields (figs. 21-22).

The future tailgate for panel 2 was graded and tracked to serve as the transport route. Its track was connected with the belt-takeup track in the belt entry by means of a slanted crosscut and a track switch. The staging room for the new face was turned 80 feet outby the originally projected location because the development entries struck solid rib-to-rib coal balls in the chain pillar at the point of projection.

Wire mesh of the chain link fence-type, in 5-foot sections, was threaded over the canopies starting at a distance of 20 feet (approximately 10 cuts) from the recovery room. Canopy extensions were dropped to facilitate handling of the wire mesh. The wire-mesh sections were spliced together with steel clips (hogrings) and 3/16-inch aircraft-type blank wire rope, leaving

FIGURE 21. - Props in recovery room.

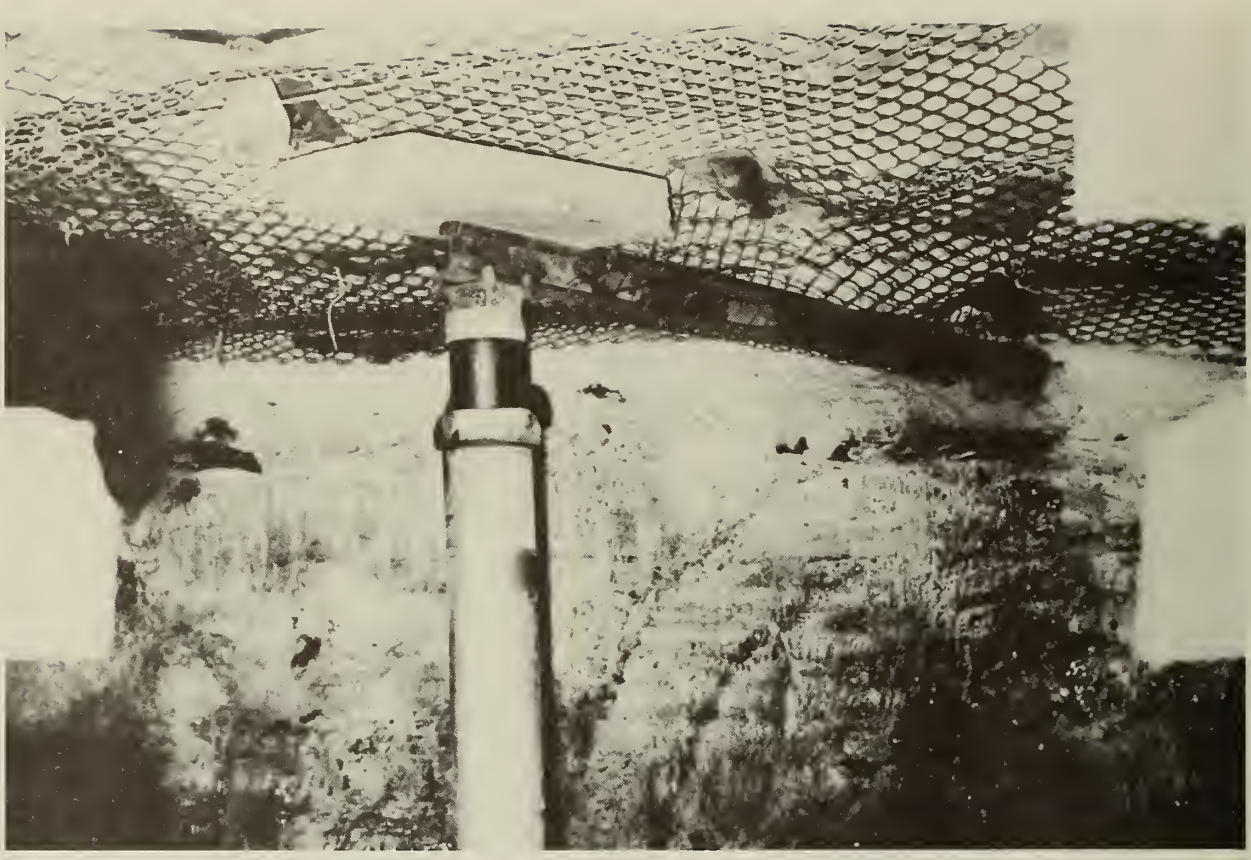


FIGURE 22. - Pinrail placed in shale roof.

6-inch overlaps. The wire mesh formed a protective mat under which the recovery could take place unimpeded by debris from the caved roof rock (fig. 23).

As expected, the 40-pound rails installed in the roof shale above the coal and supported by hydraulic jacks under the outby ends came to rest on the canopies when the shields approached the recovery room. The rails helped to secure the roof when the face holed through into the recovery room and approximately 75 shields (Nos. 20 to 95) in the midsection and tail end of the shield line yielded slightly; when the shields were advanced to the bolted roof of the recovery room, roof control was regained. The shields sustained the roof pressure, which stabilized and shifted to the recovery chain pillars and the barrier pillar, according to stress meter readings. Coal and rock sloughed from the rib of the recovery room and the floor heaved and pushed

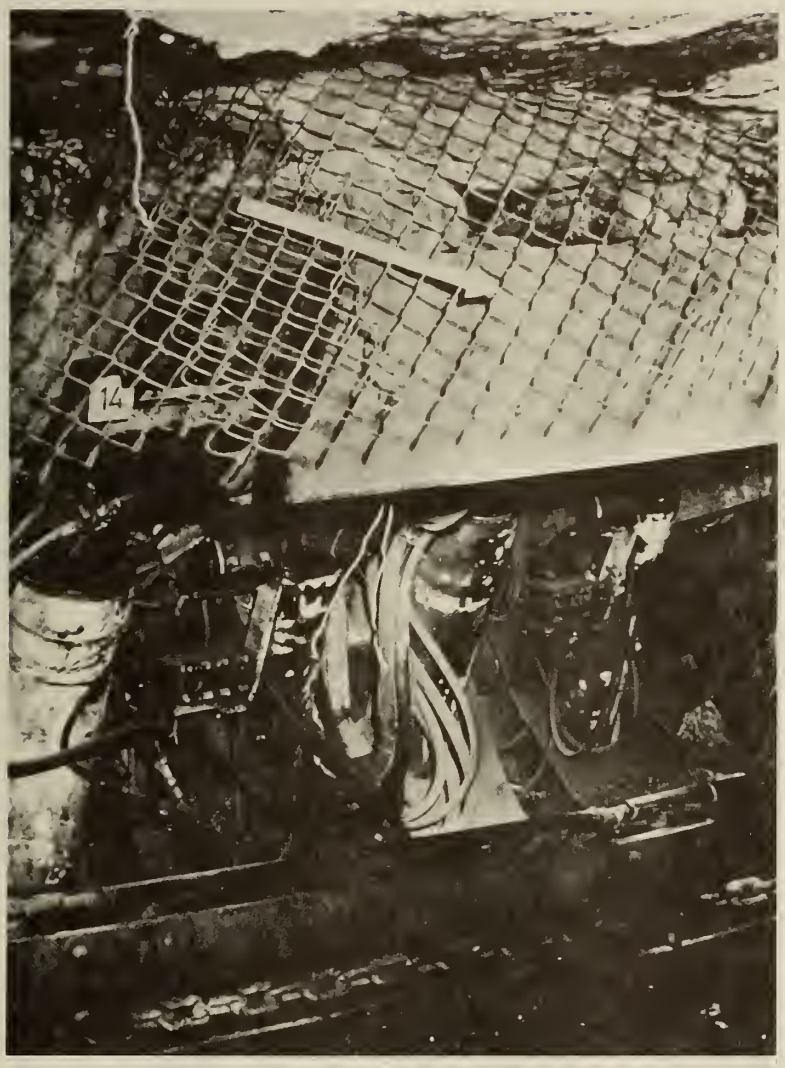


FIGURE 23. - Wire mesh laid on canopies.

the track up. Coal and debris had to be loaded out, the track removed and floor rock taken up to provide adequate clearance for shield recovery (figs. 24-25).

The shearer was run to the supply gate and, after drums and ranging arms were taken off, lifted from the pan line by two S & S Unatrack scoops. Chain, cables, and pan line in lengths of three pans were removed from the recovery room and reassembled in the face 2 staging room. Shearer and conveyor drives were taken to the surface for overhaul. The track was then relaid in the recovery room to facilitate removal of the shields.

Machinery to drag a 17.5-ton shield on the mine floor was not available; therefore, each unit, refitted with longer hoses and controlled from the adjacent unit, was advanced from the shield line, step by step, under its own hydraulic power. The advancing ram was chained to a beam that was held down by hydraulic jacks or anchored by a front-end loader or, in the case of heavy roof pressures, by a carhaul (fig. 26). When on the track of the recovery room, the shield was placed on a skid, disconnected from the hydraulic net, and pulled by the front-end loader to the headgate transfer point. The shield was then laid on its side on the rail dolly under control of the front-end loader by means of rope and sheave. (A shield in an upright position could not clear low spots in the transport route.)

Roof control during the shield recovery was maintained by a moving bulkhead formed by two shields advancing side by side along the rib side of the recovery room (fig. 27). A crib took the place of each shield removed. During the recovery operation, whenever a shield was lowered for advancing out of the shield line, five adjacent shields from Nos. 70 to 15 in the shield line yielded somewhat. Roof pressures were, at time, high in the midsection of the

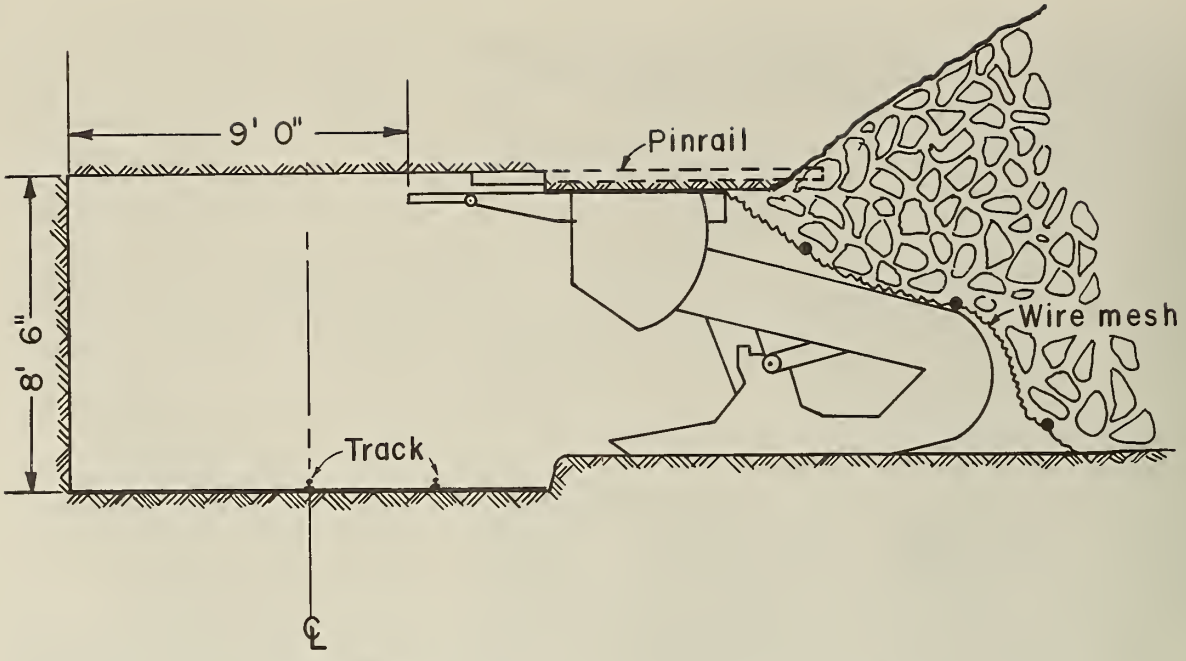


FIGURE 24. - Recovery room prior to shield recovery.

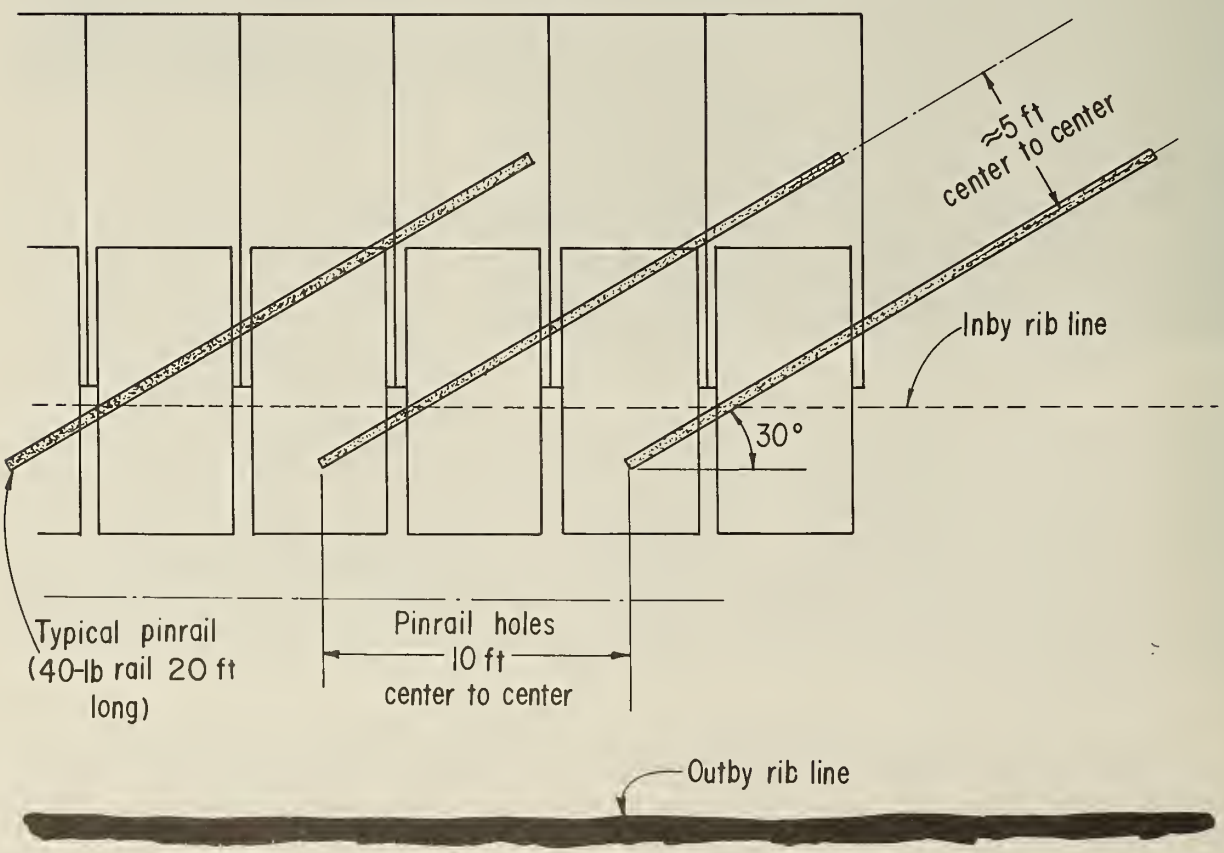


FIGURE 25. - Recovery room showing pinrail locations.

FIGURE 26. - Shield advanced from shield line during recovery.

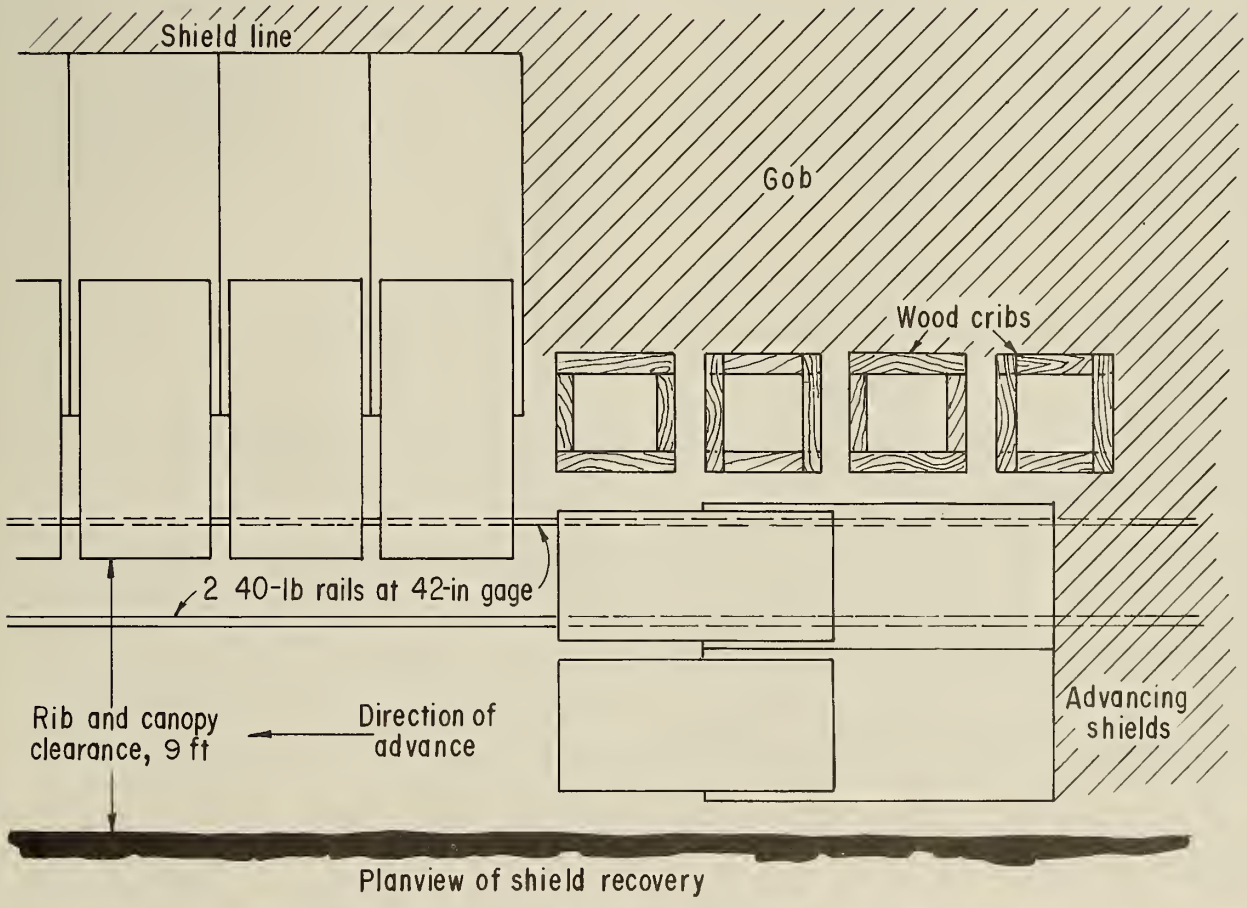


FIGURE 27. - Advancing shields forming a movable bulkhead.

shield line. The weight of the roof was on the rib side and particularly heavy at intersections with breakthroughs, even though cribs had been built across the opening. Shield No. 52 yielded to the extent that only 4-1/2 inches of leg travel was left. Each shield, lying on its side on the rail dolly, was hauled to the new face on the track by a front-end loader. After a shield reached the transfer point at the tail end of the staging area, it was placed upright and set on the staging-room track under control of another front-end loader. The shield was then maneuvered into place in the new shield line. The shields were fitted with new hoses and temporarily connected to an auxiliary power supply pending the transfer of the powerpack. The stage loader was put in its place between the conveyor head and the belt tailpiece. The shearer was introduced over the tail end of the pan line. Connecting the taildrive with the conveyor concluded the transfer of equipment. Operation of panel 2 started on August 1, 1977.

ACCIDENT EXPERIENCE

Evidently, intensive training in job safety and skills resulted in a low accident rate at the longwall when compared with that of the entire mine No. 24. Factors that contributed to the favorable results were use of roof shields, face equipment with safeguards, and a suitable mode of operation. Recovery, transportation, and reinstallation of the shields required particularly cautious planning and execution. Only one lost-time accident was reported for the period May 19, 1976, to June 19, 1977, which included installation, operation, recovery, and most of the reassembly in the new staging room. The number of no-lost-time accidents was 25. Bruises by contact with equipment and ruptured hoses were the most frequent types of accidents. Crew members near the shearer wore goggles or eye shields to prevent injury to eyes from flying debris.

GROUND CONTROL AND SUBSIDENCE STUDY

The rock-mechanics monitoring program (fig. 28) proposed and conducted by Dames and Moore was designed to achieve the following objectives:

1. To provide early-warning capability by measuring strata movement and pressures during mining so that corrective measures could be taken quickly, should an unusual situation arise.

2. To evaluate the adequacy of the roof-support system and to derive design criteria for future longwall mining efforts.

The program included the following tasks:

1. Measuring convergence in gate roads and differential rock and floor strata movement to detect bed separation.

2. Measuring in situ pressure by installing stress meters of the vibrating-wire gage type in coal and floor of the longwall panel and chain pillars (fig. 29).

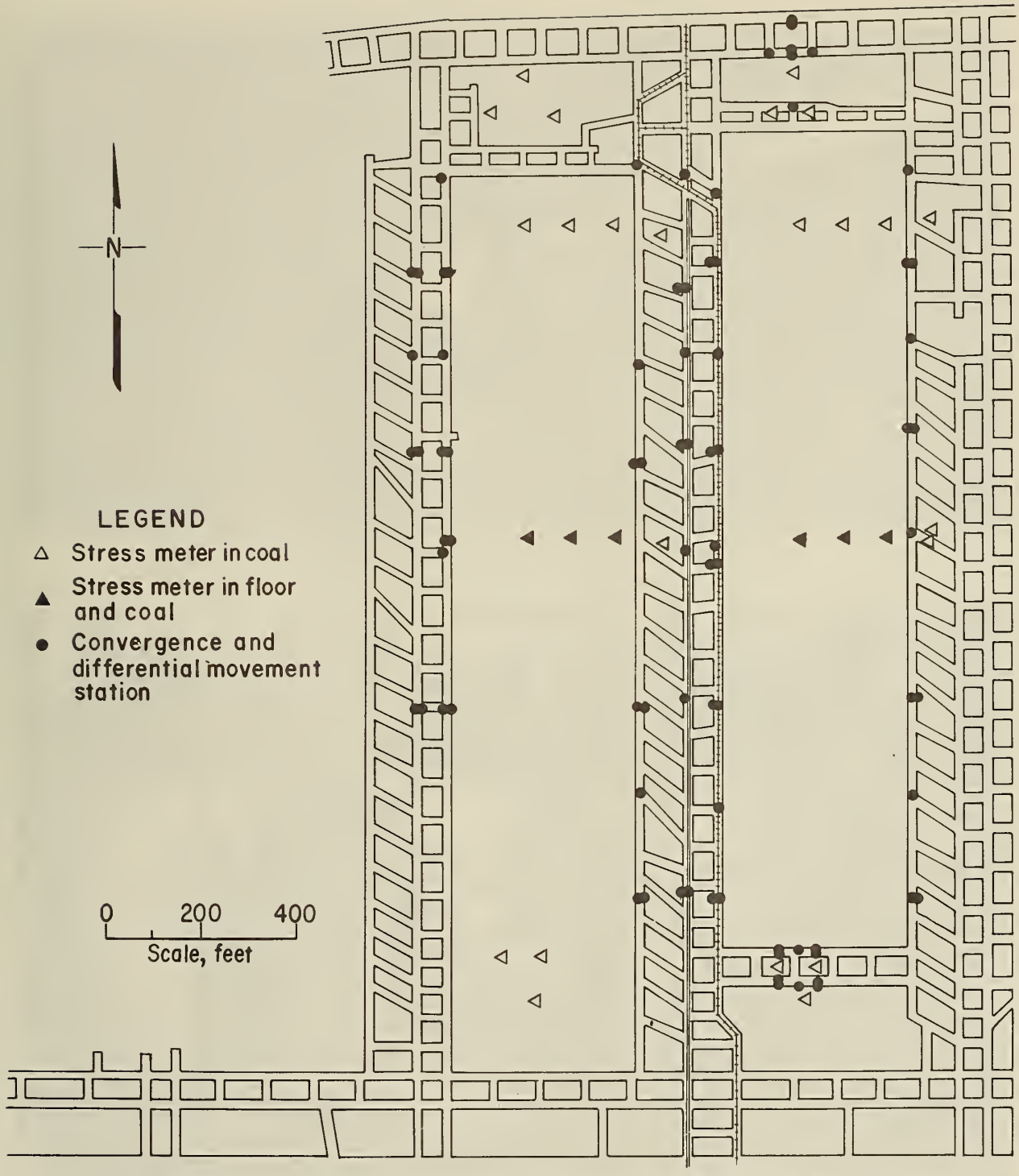


FIGURE 28. - Location of stress meters and convergence and differential movement stations in panels 1 and 2.

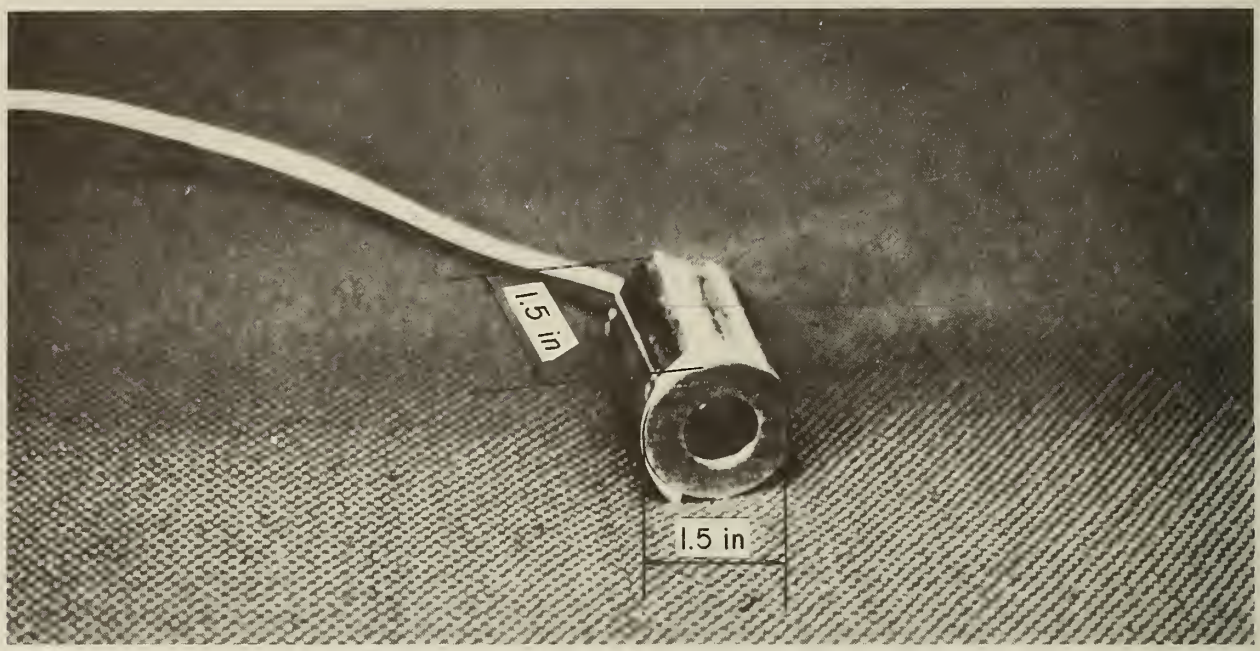


FIGURE 29. - Irad gage vibrating-wire stress meter.

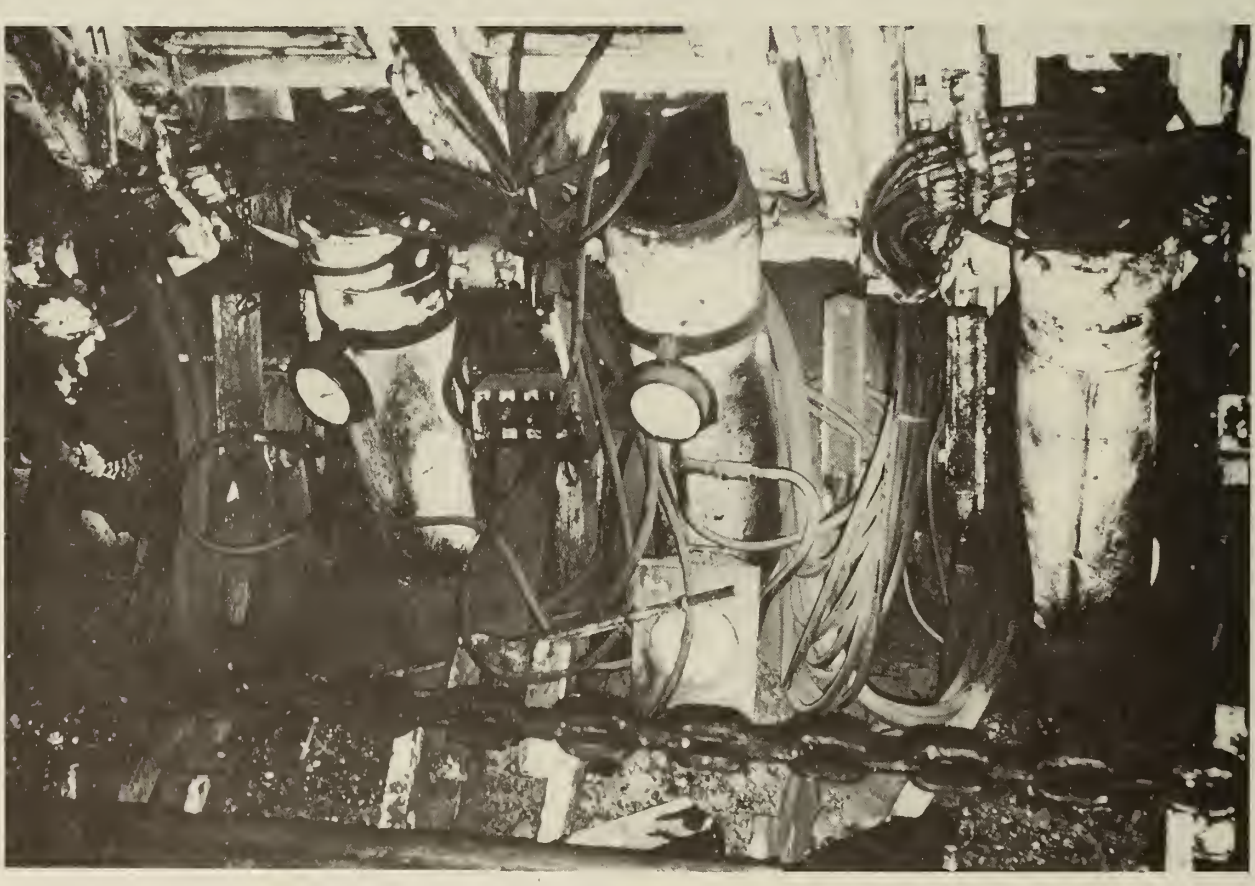


FIGURE 30. - Hydraulic pressure gages and recorder on shield.

3. Tracing the loading profile in the roof-support system by measuring hydraulic pressures with load indicators, gages, and recorders (fig. 30).

4. Determining vertical extent of caving by boreholes from surface to coalbed.

Convergence of roof and floor seldom reached or surpassed 2 inches before the face passed. Only in the tailgate a station showed a 4-inch convergence at face approach. Measurements of differential roof strata movement did not indicate bed separation between shale and limestone. Differential floor-movement readings showed slight heaving of the underclay, up to 2 inches relative to the siltstone base.

In situ pressures were measured by stress meters at the indicated locations (fig. 28). The stress meter consists of a cylindrical gage, 1.5 inches long and 1.5 inches in diameter, containing a highly tensioned steel wire mounted diametrically across the gage (fig. 29). A portable readout device vibrates the steel wire at its resonant frequency to monitor changes in the length of the steel wire which results from stress changes.

Near the staging area, the stress meter readings showed a gradual rise in stress followed by a sudden drop when the face attained a distance of 150 feet and the first major roof caving occurred. Afterwards, the stress increased gradually.

In midpanel, stress meters sharply responded to the face approach, one of them indicating a gain of 1,600 lb/sq in, before destruction.

Stress meters in the siltstone floor under the longwall panel showed a small increase at face approach, a drop to below setting pressure after the face rolled over them, and a slow gain back to setting pressure when the face moved away.

In the recovery room, stress meters indicated a sharp rise when the face cut through into the recovery room.

Pressures in the hydraulic system of shields and props were measured to compute the roof loads. Shield loads increased during idle days by approximately 10 lb/sq in. A sharp decrease in canopy loads occurred immediately before and following the first major roof falls simultaneously with the stress relief measured by stress meters near the staging area.

The instrumented hydraulic props were adequately blocked by wood headers and steel plates to sustain heavy loads. The only two jacks that yielded were the instrumented ones in the recovery room when the face cut through.

The successive caving of roof strata over the mined-out area was monitored by Time Domaine Reflectometry, as follows: A 7/8-inch coaxial cable was grouted into a NQ-size (3-inch) borehole to a depth of 670 feet. The borehole intersected the No. 6 coalbed, which is 8.7 feet in thickness, at a depth of 619 feet. A Tectronix 1502 Time Domaine Reflectometry cable tester, which

detects cable faults and breaks by Radar, was used to monitor the condition of the cable. Prior to installation, the cable is crimped at 10-foot intervals to aid in calibration and measurement. Figure 31 plots the progress of caving in terms of face advance.

After a license of right-of-way was secured from the Army Corps of Engineers, which holds the surface rights, personnel from Old Ben Coal Co. and the Bureau of Mines Denver Mining Research Center initiated surface subsidence studies over the longwall mining area to measure vertical and horizontal surface movements, including tilting of monuments (steel pipes driven into the ground). Surface strains between selected monuments are measured by extensometer or strain gages. The shape of the subsidence trough and the angle of draw are to be determined, to indicate whether the mining will affect surface structures. A total of 196 monuments were installed in 3 lines (fig. 32). Two of the lines were north-south centerlines over two adjacent longwall panels, and one was an east-west crossline. Several monuments had to be installed in boggy ground and were specially designed for the purpose.

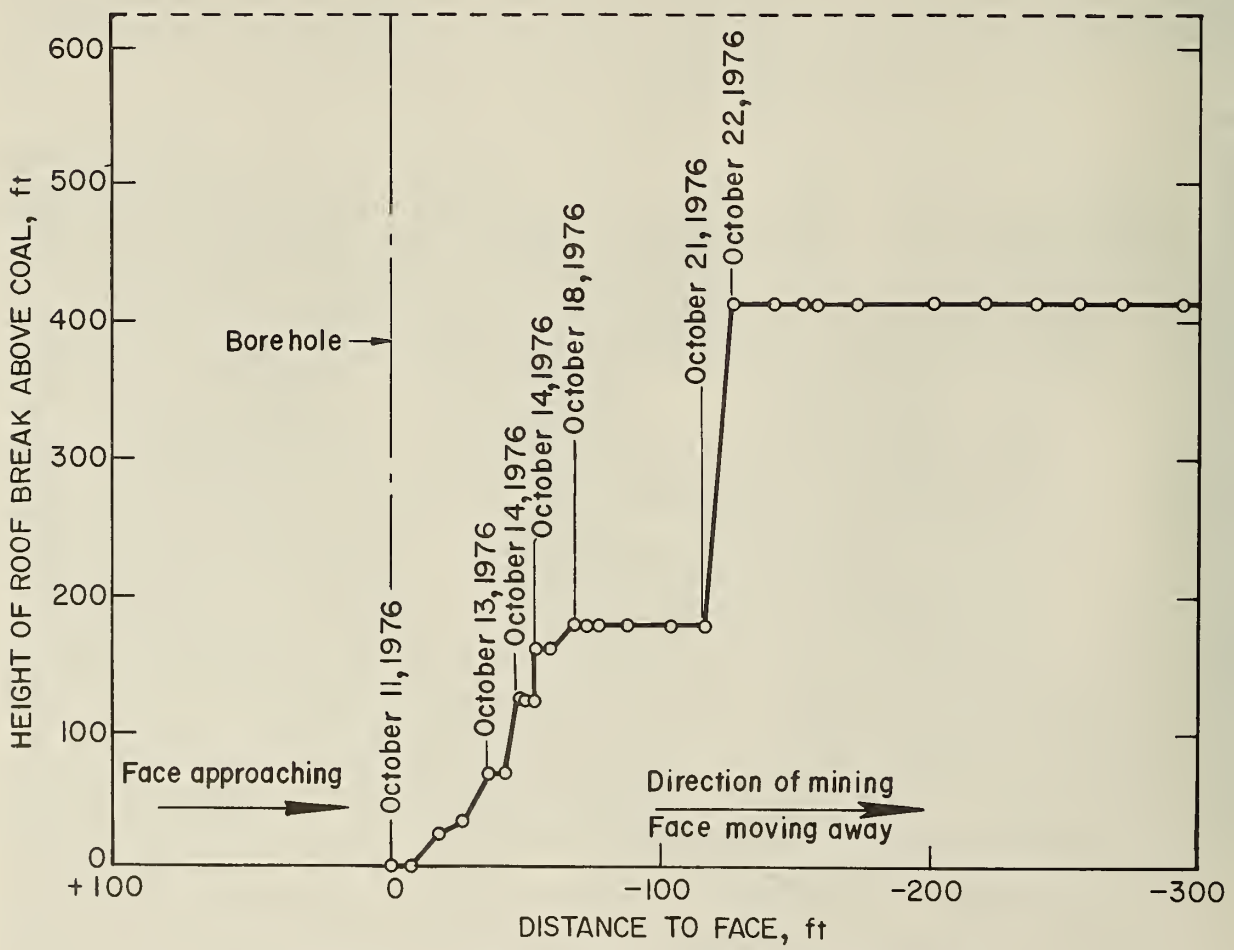


FIGURE 31. - Strata failure in terms of face advance.

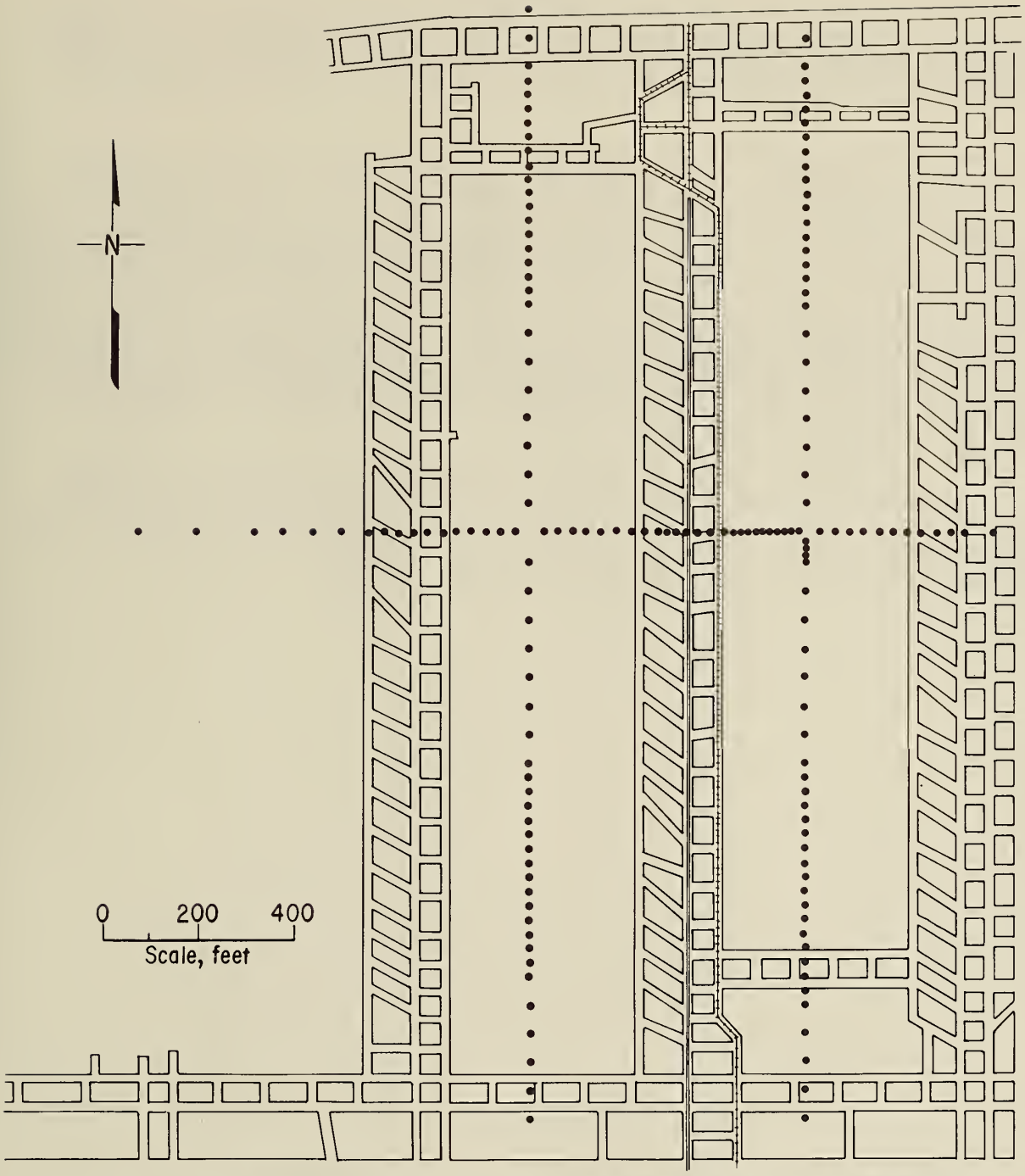


FIGURE 32. - Map showing location of 196 monuments installed at the mine surface.

The Illinois State Geological Survey was also involved in providing early warning of unusual conditions by mapping the longwall panel area as it was developed and mined. The Survey explored the adjacent strata by vertical boreholes up to 40 feet deep.

CONCLUSIONS

It is premature to draw final conclusions because only one-third of the program has been completed. The low accident rate is an indication that the longwall system may be safer than room-and-pillar mining. The use of shields appears to be the major factor contributing to successful roof control. Short spans of exposed roof, quickly supplied support, stability of structure, and full shelter for face crews are some of the advantages that led to the introduction of shields. The characteristics of the selected shield satisfied criteria by deploying the highest mean load density yet attained for any shield, vertical travel of 4 feet in one hydraulic stroke, and a base designed to overcome obstacles on the floor.

The rock-mechanics study indicated the extent of caving in the strata profile, but did not reveal any damaging rock-mass movements or loads. Geological mapping by the Illinois State Geological Survey contributed to the knowledge of the area and pointed to possible hazards.

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